TN
1
A5
Vol. 88
N/C

TRANSACTIONS

OF THE

American Institute of Mining and Metallurgical Engineers

(INCORPORATED) and Petroleum

COAL DIVISION
1930

CONTAINING PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD IN NEW YORK, FEBRUARY, 1928, FEBRUARY, 1929 AND FEBRUARY, 1930

NEW YORK, N. Y.
PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
1930

PRINTED IN THE UNITED STATES OF AMERICA

THE MAPLE PRESS COMPANY, YORK, PA.

PREFACE

With this volume the Coal Division of the A. I. M. E. makes its bow to the public. Not that coal is a new subject for discussion at Institute meetings, but that the new form of organization and publication permits it to be given the attention that its importance warrants. At the first meeting of the Institute, nearly 60 years ago, at Wilkes-Barre, R. P. Rothwell presented a paper on Waste in Coal Mining. This with 10 other papers on coal was printed in the first volume of TRANSACTIONS and in the long series of volumes published since have appeared many of the most important papers that have been printed regarding the technology of coal production and utilization. For many years the programs on coal were in charge of a Technical Committee, but in February of the current year this was expanded into a Division with its own by-laws and elected officers. Under the Institute form of organization a Technical Division is almost an autonomous society within the larger body. It makes its own rules, subject to approval by the Board of Directors, organizes its own programs and substantially determines which of the papers presented in the course of the year shall be republished in the special volume of TRANSACTIONS devoted to the field of the particular Divisions. Final approval of all papers rests with the general Papers and Publications Committee, in which all Divisions and Technical Committees are represented, but this committee almost invariably follows the advice of the Division.

It is now, accordingly, open to men interested in the broad field of coal classification, production, preparation and use to hold meetings, organize programs and publish papers without the expense of maintaining a separate society. This service the Institute is glad to offer to the industry. That the problems are vital and that discussion may well be profitable is sufficiently indicated by the contents of this volume. The program prepared for the first independent meeting of the Coal Division, at Pittsburgh, Sept. 11, 12 and 13, shows furthermore that interest in the field is as keen as at any time since the founding of the Institute. The cooperation of all in the field is cordially invited.

H. FOSTER BAIN,
Secretary.

FOREWORD

This volume is the first that has been prepared by the Coal Division of the A. I. M. E. and contains most of the papers that have been submitted on matters pertaining to coal before the Institute during the last three years. Under "Mining" are included six papers that were presented before the Committee on Ground Movement and Subsidence, which have been included in this volume so that all papers relating to coal or coal mining may be kept together; also, one paper which was presented before the Committee on Ventilation and three papers which were presented before the Committee on Mining Methods have been included in this volume, for the reason outlined above.

The papers on coal cleaning give results of some scientific work by the U. S. Bureau of Mines and also the results of cleaning coal by wet washing from a commercial standpoint.

The results of some tests on the coking properties of coal are given in two papers presented by the Bureau of Mines; and a paper giving the relation of the by-product coke-oven gas to the natural gas supply of the Pittsburgh district brings up a phase of the gas question that is going to be of increasing importance in future years and is of great importance now, as long gas lines are being extended in many parts of the country which will sooner or later have to depend for their ultimate success on some other source of supply than that coming from gas wells.

By far the greater part of the volume is devoted to papers on the classification of coal, which were presented at the February meetings in 1928 and 1930. Mr. Fieldner, in his introduction, has given the reason for the presentation of these papers. They include a vast amount of information on the composition and classification of coal from scientific, commercial and use standpoints. This subject has been of great importance to the industry in past years and there is no doubt of its continuing interest. It is hoped that the work of this Committee will result in the end anticipated, and the Coal Division is glad to be of service to the industry in this connection.

Howard N. Eavenson,
Chairman, Coal Division, 1930

CONTENTS

PAPERS

Mining

Cleaning

Coking

Classification

A. I. M. E. OFFICERS AND DIRECTORS

For the year ending February, 1931

President, William H. Bassett, Waterbury, Conn.
Past President, George Otis Smith, Washington, D. C.
Past President, Frederick W. Bradley, San Francisco, Calif.
Treasurer, Karl Eilers, New York, N. Y.
Secretary, H. Foster Bain, New York, N. Y.
Assistant Secretary, A. B. Parsons, New York, N. Y.

Vice-presidents

Henry Krumb, Salt Lake City, Utah
Scott Turner, Washington, D. C.
H. A. Guess, New York, N. Y.
George D. Barron, New York, N. Y.
Henry A. Buehler, St. Louis, Mo.
Edgar Rickard, New York, N. Y.

Directors

Rolland Craten Allen, Cleveland, Ohio
Cadwallader Evans, Jr., Scranton, Pa.
John M. Lovejoy, New York, N. Y.
John A. Mathews, New York, N. Y.
Milnor Roberts, Seattle, Wash.
Herman C. Bellinger, New York, N. Y.
Karl Eilers, New York, N. Y.
Herbert G. Moulton, New York, N. Y.
John V. W. Reynders, New York, N. Y.
Robert E. Tally, Jerome, Ariz.
Erle V. Daveler, Butte, Mont.
Eugene McAuliffe, Omaha, Neb.
Harvey S. Mudd, Los Angeles, Calif.
Francis W. Paine, Boston, Mass.
William R. Wright, Chicago, Ill.

Officers of the Coal Division

Howard N. Eavenson, *Chairman*
W. H. Fulweiler, *Vice-chairman*
M. W. von Bernewitz, *Secretary*

Executive Committee

Thomas G. Fear, *Chairman;* Thomas H. Clagett, George Watkin Evans, M. H. Fies, John A. Garcia, Herbert D. Kynor, E. J. Newbaker, Kenneth A. Spencer, F. G. Tryon.

Programs and Meetings

M. D. Cooper, *Chairman;* A. C. Callen, C. T. du Rell, C. Enzian, E. A. Holbrook, Gordon MacVean, Theodore Marvin, A. C. Noe, Henry H. Otto, R. G. Pfahler, H. S. Salmon, L. H. Schnerr, F. G. Tryon, William R. Wright.

Bituminous Coal Production

Frank Haas, *Chairman;* R. L. Adams, C. S. Blair, George Watkin Evans, John A. Garcia, H. S. Geismer, Ivan A. Given, P. C. Graney, C. G. Hall, Carl T. Hayden, Chester M. Lingle, E. P. McCrorken, G. C. McFadden, W. S. Moore, C. C. Morfit, Thomas Moses, H. E. Nold, James W. Paul, George S. Rice, J. D. Rogers, Cecil W. Smith, P. C. Thomas, Jerome White, C. C. Whittier.

Anthracite Coal Production

E. H. Suender, *Chairman;* R. H. Buchanan, D. J. Carroll, S. D. Dimmick, Edward Griffith, J. C. Haddock, E. P. Humphrey, A. B. Jessup, J. Latimer Lee, R. V. Norris, Jr., George Roos, H. M. Smyth, B. H. Stockett, Arthur Waldman, J. B. Warriner.

Evaluation of Coal for Blast-furnace Coke

J. R. Campbell, *Chairman;* A. C. Fieldner, F. A. Jordan, C. E. Lawall, C. E. Lesher, Clarence E. Meissner, John B. Price, Harold J. Rose, H. W. Seyler, E. H. Shriver, W. T. Thom, Jr., Howard P. Zeller.

Uses of Coal

W. H. Fulweiler, *Chairman;* W. D. Barrington, W. S. Blauvelt, John B. Corrin, L. E. Geohegan, O. E. Kiessling, Henry Kreisinger, E. H. Lawall, G. E. Lyman, George A. Orrok, J. D. Sisler, D. P. Smith, J. R. Van Pelt, Jr., F. R. Wadleigh, Samuel S. Wyer.

Coal Preparation

J. B. Morrow, *Chairman;* Andrews Allen, Ray W. Arms, George E. Bayles, Byron M. Bird, F. H. Blatch, Thomas M. Chance, Joseph Daniels, G. R. Delamater, Thomas Fraser, Robert E. Hobart, D. R. Mitchell, E. O'Toole, R. E. Rightmire, H. M. Search, R. H. Sherwood, Humphrey D. Smith, Paul Sterling, G. M. S. Tait, Howard J. Thomas, G. V. Woody, H. F. Yancey.

Machine and Strip Mining

Eugene McAuliffe, *Chairman;* R. H. Allport, S. H. Ash, R. G. Baughman, S. W. Blakslee, Edward Bottomley, Harry Boyle, A. F. Brosky, F. E. Butcher, C. E. Dobbin, Cadwallader Evans, Jr., Edward Graff, C. C. Hagenbuch, H. F. Hebley, Otto Herres, Jr., F. V. Hicks, John K. Holmes, W. K. Kavanaugh, F. S. McConnell, W. S. Rausch, C. F. Spencer, E. F. Stevens, S. A. Taylor, R. Y. Williams, L. E. Young.

Safety

Thomas G. Fear, *Chairman;* J. T. Ryan, *Vice-chairman;* F. E. Bedale, M. H. Fies, J. J. V. Forbes, Daniel Harrington, W. B. Hillhouse, J. M. Jennings, John E. Jones, R. M. Lambie, G. S. McCaa, C. McDowell, J. J. Rutledge, F. J. Stortz, D. A. Stout, W. C. Stratton, Thomas D. Thomas.

Representing Division on Papers and Publications Committee

R. Dawson Hall, Cadwallader Evans, Jr., R. V. Norris, Jr., E. W. Parker, Harold J. Rose.

Representing Division on Coordination of Research Committee

J. S. Miller, G. St. J. Perrott.

Ventilation Problems at the World's Largest Coal Mine

By Henry F. Hebley,* Chicago, Ill.

(New York Meeting, February, 1930)

The New Orient mine, owned and operated by the Chicago, Wilmington & Franklin Coal Co., has caused a great deal of comment and interest because of its unusual features and huge daily production. It is in Franklin County, Illinois, adjacent to the town of West Frankfort. The seam being mined is the Illinois No. 6 which in this locality is 9 to 12 ft. thick and lies approximately 500 ft. below the surface. Without fear of contradiction, it can be said that this is one of the largest coal mines in the world from a daily production standpoint. During 1928, New Orient produced 2,400,000 tons, which amounts to over 10,000 tons each day the mine operated that year. The mine is built for a capacity of 12,000 tons but has actually hoisted over 15,000 tons in one 8-hr. shift. This enormous production can perhaps be better visualized if it is realized that each day's operation depletes over two acres of coal.

The general features of this mine have already been fully covered in a paper by George B. Harrington,[1] President of the Chicago, Wilmington & Franklin Coal Co. In a paper of the nature of Mr. Harrington's, it was unnecessary to stress any particular phase of mining conditions, consequently the ventilation problems were touched upon but lightly. In this paper an attempt has been made to describe some of the problems which have been encountered in providing adequate ventilation for the mine.

Ventilating Conditions

The acreage allotted to New Orient mine has a length, north and south, of approximately 6 miles, and a width, east and west, of 2 to 4 miles. A number of important considerations caused the hoist shaft, air shaft and surface plant to be located near the southeast corner of the property, instead of near the center, as is usual. During the initial phase of the mine development, the mine was ventilated through these shafts, although there was a full realization that at some future date other shafts would be required.

---

* Mechanical Engineer, Allen & Garcia Co.

[1] G. B. Harrington: New Orient, an Unusual Coal Mine. *Trans.* A. I. M. E (1925) **72**, 798.

The southern half of the acreage (Fig. 1) is being developed from two sets of main north headings, each consisting of two intake and two return airways with the haulage roads on the return air, as is customary in the

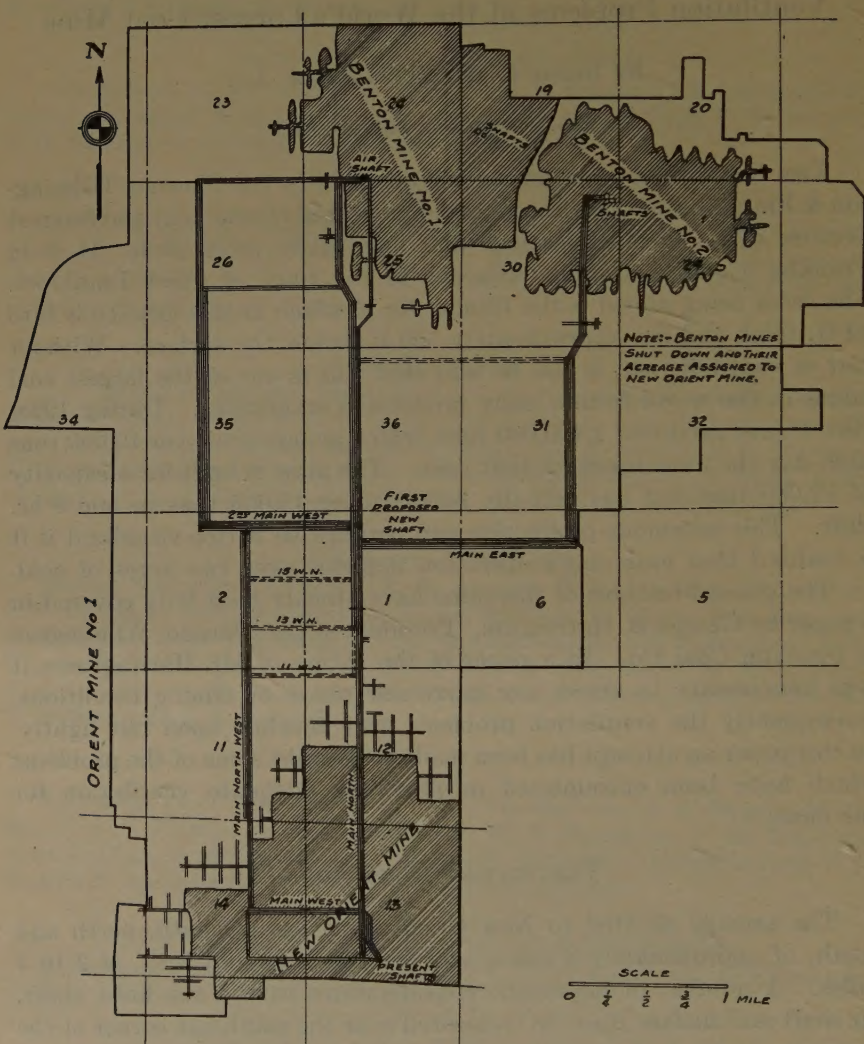


FIG. 1.—PROPOSED METHOD OF VENTILATION, NEW ORIENT MINE.
Working extended to July 1, 1928.

Illinois coal field. The wider north half of the property will probably be worked from three sets of parallel main entries.

At present from each set of main entries, cross entries are driven at intervals and from these, in turn, room entries are driven. The panel

room-and-pillar system is used, each panel being sealed immediately after mining has ended.

The ventilating equipment originally installed at the mine had a rating as follows:

Fan, 12 ft. dia. by 5 ft. face, normally blowing emergency exhausting
Motor, 500 hp.
Normal capacity, 400,000 cu. ft. per min.
Static pressure, 5 in. water gage
Speed, 185 r.p.m.
Brake horsepower, 420
Mechanical efficiency, 75 per cent.
Volumetric capacity, 382 per cent.
Equivalent mine orifice to pass 400,000 cu. ft. per min. at 5 in. w. g., 71.4 sq. ft.
Hoist shaft, two skip-hoist compartments on return air.
Air shaft, semielliptical; two-man and material cage compartments on return air, and 130 sq. ft. area intake-air compartment

A coal roof is left throughout the mine and only a moderate amount of timbering is required.

Average entry dimensions affecting ventilation are:

Clear area of main entries, approx. 12 by 8, sq. ft.	95
Clear area of main entries, developing, approx. 11 by 7 ft. 6 in., sq. ft.	83
Perimeter of main entries, ft.	40
Perimeter of main entries, developing, ft.	36
Number of main air splits	3
Total mine methane generation, cu. ft. per min.	1100 to 1400

Ventilating Problems

As development progressed from the time the mine was opened, the demand on the fan became heavier until in July, 1926, it was necessary to relieve the resultant air conditions. The fan, when tested at that time, was found to be working as follows:

Capacity, cu. ft. per min.	220,000
Static pressure, inches, w. g.	4.23
Speed, r.p.m.	190
Brake horsepower	228
Mechanical efficiency, per cent.	50
Volumetric capacity, per cent.	205
Equivalent mine orifice, sq. ft.	48.6

These results led to an investigation to ascertain whether it was the opportune time to drive the two additional air entries that had been contemplated and thus relieve the throttling action that was affecting the ventilating system. It was found that the air was carried 1800 ft. from the downcast-shaft air compartment to the main air split, through two entries with a combined area of 120 sq. ft. A throat of small area,

immediately beyond the sharp right-angled turn at the bottom of the air shaft, also restricted the air flow.

After the test it was felt that more cross-sectional area was required and conditions were temporarily remedied by driving the two intake entries mentioned (making a total of four) from the downcast shaft to the junction of the main north and main west entries. However, the continual development of the mine after that time brought about a condition in the latter part of 1928 that caused the ventilation again to become inadequate and inefficient. With a water gage of 5 in., a volume of only 300,000 c.f.m. was entering the mine.

When it is realized that 1600 men are employed underground, that the mine produces an enormous daily tonnage, that the main headings are lengthened by approximately ½ mile each year and that these entries will eventually be over 6 miles long, the necessity for broad and ample ventilation provisions over the life of the mine will be understood. It was felt that the time had arrived when the possibility of a new shaft, or shafts, should receive serious consideration, so with the object of providing for the entire future life of the property a thorough study was made to find out the best and most economical methods whereby this could be accomplished. The first calculations quickly proved that in no practical way would it be possible to mine out the assigned acreage with a four-entry system, using only the existing hoist and air shafts.

This fact established, the main problem resolved itself into two closely related questions:

1. If a new shaft (or shafts) is necessary, where is the best location and should it have both upcast and downcast or be a single-compartment shaft?

2. Is it possible or desirable to continue the present four-entry system on main headings, or would it be better to change to five, six, or more entries in all future development?

Analysis of Problems

Before attempting any detailed ventilation calculations, it was necessary to establish values for certain factors which are usually constant for the airway conditions in any one mine. A great deal of care was used in determining the friction factor K, as this had a material influence on the calculations, especially where it was realized that the old established factor given by various authorities differed by as much as several hundred per cent. It was decided to base the calculations on a factor determined by actual mine conditions; this was carried out using the fundamental formula:

$$P = \frac{KSV^2}{A} \text{ lb. per sq. ft.} = \frac{KSV^2}{5.2A} \text{ in. water gage,}$$

where P = drop in pressure in inches of water,
K = friction factor,
S = rubbing surface in square feet,
V = velocity in feet per minute,
A = cross-sectional area of air course in square feet.

The factor $\frac{SV^2}{A}$ for various sections of the air course, based on readings and dimensions taken in the mine, is as follows:

Shaft	2,075,000,000
Air mains	698,000,000
Main north, MW-3W	1,042,000,000
3W-5E	724,000,000
5E-5W	117,500,000
5W-7E	332,000,000
7E-7W	50,000,000
7W-9E	127,200,000
9E-9W	14,100,000
9W-11E	17,300,000
Development of entries, 4	7,830,000
Development of entries, 2	42,100,000

$$\text{Total } \frac{SV^2}{A} = 5{,}247{,}000{,}000$$

$$P = \frac{K}{5.2} \times \frac{(SV^2)}{A} \text{ from which } K = \frac{5.2P}{(SV^2/A)}$$

As the water gage registered was 5.0 in.,

$$K = \frac{5.2 \times 5.0}{5{,}247{,}000{,}000} = \frac{26.0}{5{,}247{,}000{,}000} = 0.000{,}000{,}00496 =$$

$$49.6 \times 10^{-10} = \text{friction factor.}$$

Value assumed = 50×10^{-10} in main entries.

For smooth-lined shafts, the friction factor K was taken as 27×10^{-10}.

The value of K is in close accordance with recent values determined by experiment by investigators of the U. S. Bureau of Mines,[2] and the University of Illinois Engineering Experiment Station.[3]

After determining the basic constant factors, several charts and tables were made, similar to Fig. 2, which covered all possible air quantities entering into the problem and considerably shortened the onerous mathematical work attached to each of the numerous cases worked out.

The results of the calculations are plotted on the ventilation diagram, Fig. 3. On this chart the ordinates represent the mine resistance in inches water gage, and the abscissas represent time intervals in years. Six critical periods are shown, at each of which conditions have changed from the preceding period because of the projected progress and extension of the mine workings. During these years the mine workings are

[2] U. S. Bur. Mines *Bull.* 285 and 261.

[3] Univ. Ill. Eng. Expt. Sta. *Bulls.* 158, 170, 184, 199.

gradually changing from one ventilating condition to another, therefore straight lines have been drawn between the points as representing the average change, although this is only approximately true. A brief description of the mine conditions at each period follows:

1929.—Initial conditions: fan delivery, only 300,000 cu. ft. per min. at 5 in. water gage; four-entry system; three air splits, main north territory, main northwest territory, and separate split to southwest territory.

1930.—New air shaft sunk and connected; mine workings otherwise as above.

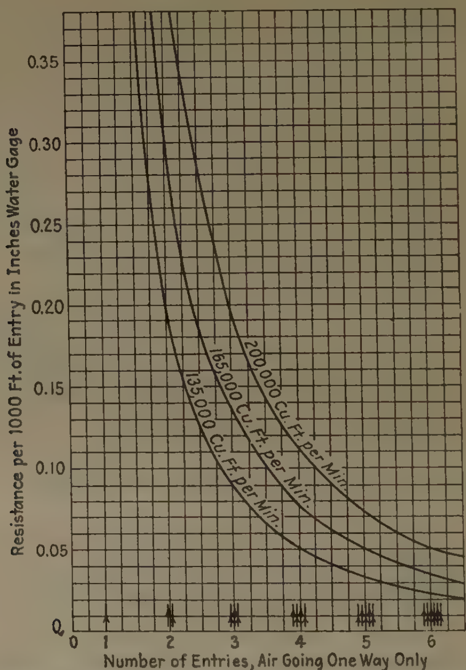


FIG. 2.—MAIN ENTRY RESISTANCE PER 1000 FT. AT VARYING QUANTITIES OF AIR.
Number of entries refers to either intake only or exhaust only.
Area of each entry, 95 sq. ft.; perimeter, 40 ft.; value of K, 50×10^{-10}.

1931.—Main northwest entries connected to new shaft.

1932.—Second main west driven and connected to workings on main northwest.

1933.—Territory on southwest split finished, room territory concentrated on main north and main northwest, but all room territory south of new air shaft and second main west. In the five years between 1933 and 1938, the mine workings gradually pass north of the second main west.

1938.—All working territory north of new air shaft and second main west. An alternate case is also considered where the northwest territory

is worked at a slower rate to offset the higher resistance of this section and permit a more or less natural air split.

1942.—Entries assumed connected to one or both of the shafts at the Benton mines. Either two or three-way development of workings in north half of acreage.

The remaining life of the mine will not introduce any further important changes.

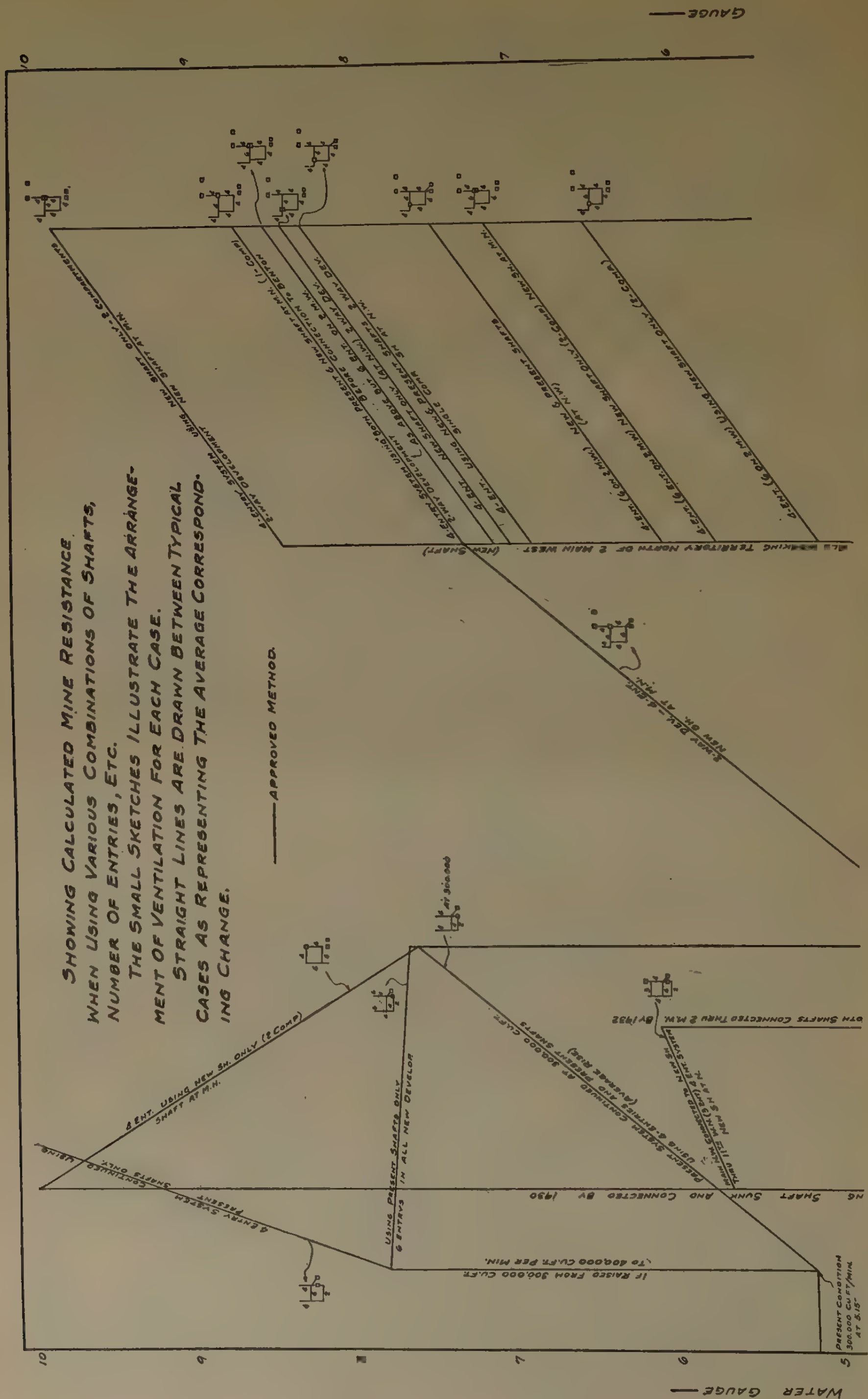
Shaft Location

The results of the calculations which are illustrated by the ventilation diagram, Fig. 3, showed that one of the new air shafts was necessary immediately, regardless of any possible practical combination and number of main entries. This point decided, the best location was next considered. With due regard for the future development of the mine, as well as best ventilation and power conditions, the shaft was tentatively located at different points and the resistance of the mine figured for various years as development proceeded.

It became apparent after a number of trials that it should be located in the middle of the property somewhere along the course of the projected main east or second main west entries. Entering into the calculations for each location was the fact that the mine might be worked in three directions after reaching the center of the property or possibly in only two directions for a period of time; also, that the air shaft at old Benton No. 2 might not be usable.

After repeated trials, it was found that locating the shaft at the junction of the main north and second main west would give the speediest relief to the mine's ventilating conditions, provided that either the eleventh, thirteenth or fifteenth west-north cross entries were driven through to the main northwest by the time the new air shaft was completed, inasmuch as the main northwest entries would control the mine resistance, because of the much longer course of air travel. This also had an advantage in that the main north entries could be driven to the shaft site in about the time necessary to sink this shaft. However, a location at the junction of the main northwest and second main west would give better results for a considerable time, especially if a two-way development was to be undertaken in the north half of the acreage. The location for the first shaft was finally chosen at the junction of the main east and main north headings, as shown on Fig. 1, because of the numerous advantages of this site as shown by the calculations, the results of which are diagrammatically illustrated on Fig. 3.

Closely interrelated to all of the problems concerning the location of the shaft was the question of its design. Due consideration of the air flow and resistance eventually led to the conclusion that both an upcast and downcast were desirable, whether included in one or two shafts.


SHOWING CALCULATED MINE RESISTANCE
WHEN USING VARIOUS COMBINATIONS OF SHAFTS,
NUMBER OF ENTRIES, ETC.
THE SMALL SKETCHES ILLUSTRATE THE ARRANGEMENT OF VENTILATION FOR EACH CASE.
STRAIGHT LINES ARE DRAWN BETWEEN TYPICAL CASES AS REPRESENTING THE AVERAGE CORRESPONDING CHANGE.
APPROVED METHOD.
WATER GAUGE
GAUGE

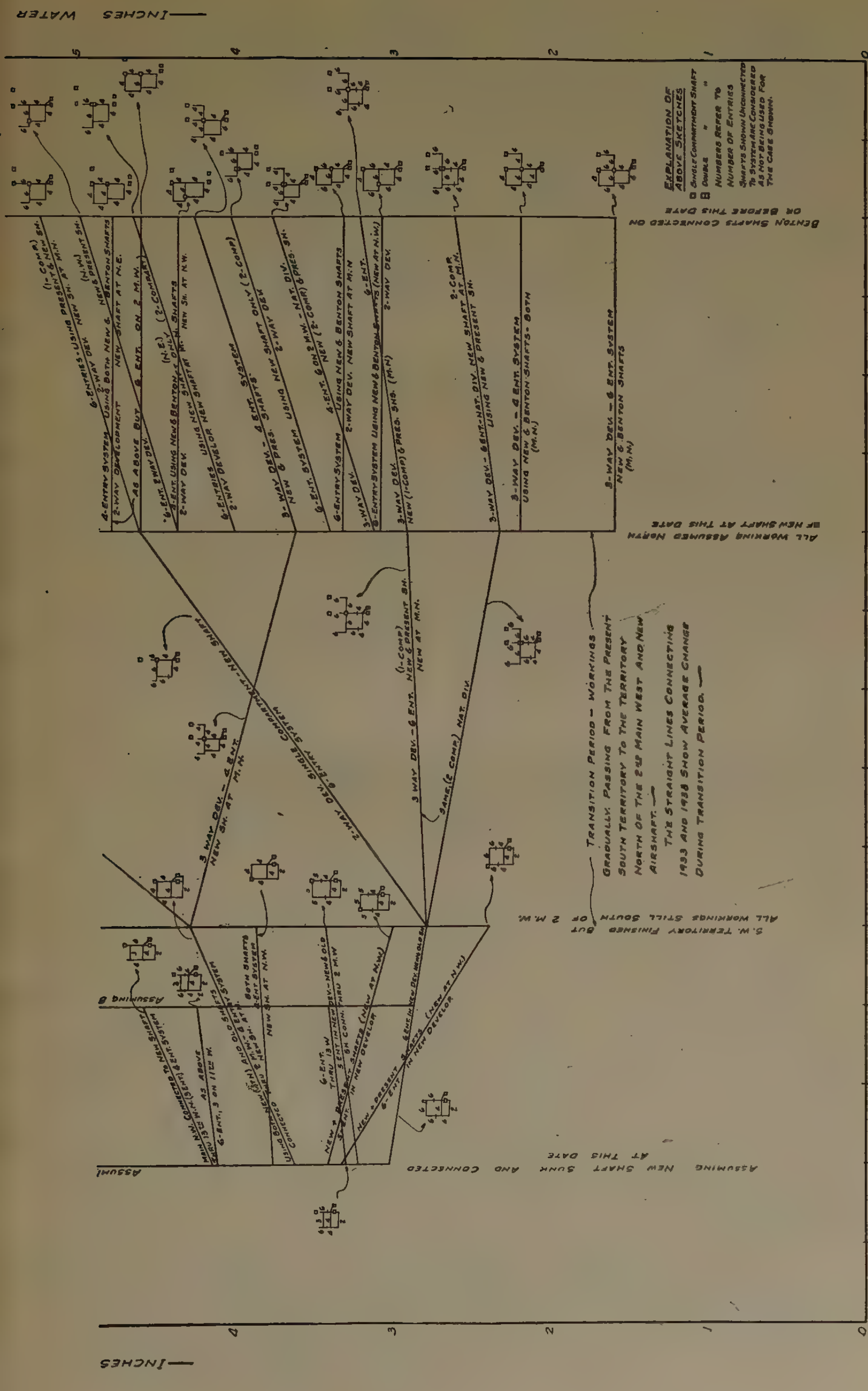


FIG. 3.—VENTILATION DIAGRAM, NEW ORIENT MINE.

Designs and estimates of the various types of shafts were then made, consisting of rectangular wooden double-compartment, rectangular wooden single-compartment, concrete elliptical types with and without partitions and circular concrete shafts.

Taking into account the estimated cost, the fire hazard, the fact that if two separate shafts were adopted, expenditure of the second one could be deferred three or four years and that the location of the second shaft could be modified to give the most advantageous arrangement for connection to the mine airways, two circular concrete shafts were adopted. The circular cross-section of this type of structure also keeps the rubbing surface down to a minimum.

Economical Shaft Design

The various questions influencing the dimensions of a shaft, or shafts, of economic design were studied at length and calculations were made giving due consideration to all the factors affecting the proportions of the structure. Computations of the estimated capital expenditure were carried out for shafts of various cross-sectional areas, together with calculations showing the corresponding influence on the annual power cost.

Assuming a life of 20 years for the shaft, interest on investment at 7 per cent., and power at 1.5 c. per kw-hr., the total investment was capitalized and, with due allowance for depreciation, the figure was brought down to a present-worth value. The operating costs, consisting mainly of power, as it was assumed that the superintendence, oil, waste, etc. would be the same in all cases, were considered as applicable only during the life of the shaft; namely, 20 years. Consequently, after obtaining the capitalized value of the annual operating cost, the present worth of this capitalized value was deducted from it, thus giving the equity. The sum of the present worth of the capitalized investment cost and the equity, brought to an annual basis, gave the annual total cost.

These amounts calculated for shafts of various cross-sections gave the requisite figures from which were plotted as ordinates points on the curve shown on Fig. 4. The abscissas consisted of shaft cross-sectional area. A study of this curve shows a diminution in total annual cost with the increase of shaft area until 158 sq. ft. is reached; thereafter, the cost becomes greater with any further enlargement in cross-sectional area. Two standard diameters have been marked on the curve and as the smaller or 14-ft. diameter gave the more economical area, it was adopted.

Drawings of the shaft are shown on Figs. 5 to 8 which illustrate clearly the method of construction. For safety, it was deemed advisable to have an emergency means of egress at the new shaft, but on account of the circular shape of the shaft, the cost of a stairway and other pertinent

reasons, it was thought best to provide an emergency man-cage running on rope or wood guides. Normally, the cage would be swung clear of the shaft by means of a headframe similar to a stiff-leg derrick, but when necessary the cage would be quickly brought into position and lowered in the shaft.

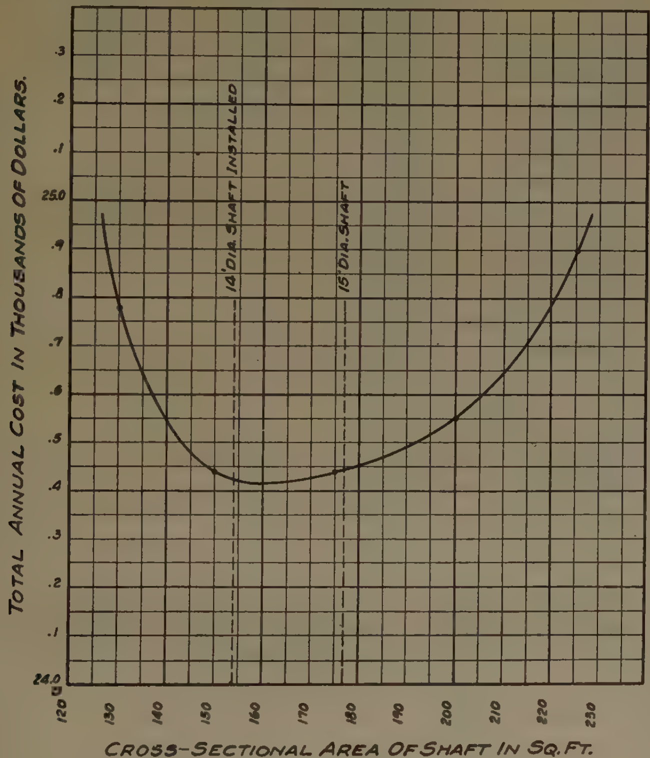


FIG. 4.—MOST ECONOMICAL CROSS-SECTION FOR AIR SHAFT.

Fig. 9 illustrates the layout of the mine at the bottom of the first of the new air shafts and shows the connections of the airways to it. The main north entries have been designated main north A, B, C, D, E and F, of which A and B are haulage and the remainder are air courses. For the present this new air shaft will be used as an upcast. The air delivered by the present fan will have a volume in the neighborhood of 400,000 cu. ft. per min. and air will be discharged from both this new shaft and the present main shaft. Connection was made between the shaft and the workings on Jan. 11, 1930, and based on readings taken on Feb. 4, 1930, the percentages of air in the various shafts were as follows:

	Cu. Ft. per Min.
Volume of air in downcast air shaft	297,000
Volume of return air in main shaft	195,000
Volume of return air in the new air shaft	105,000

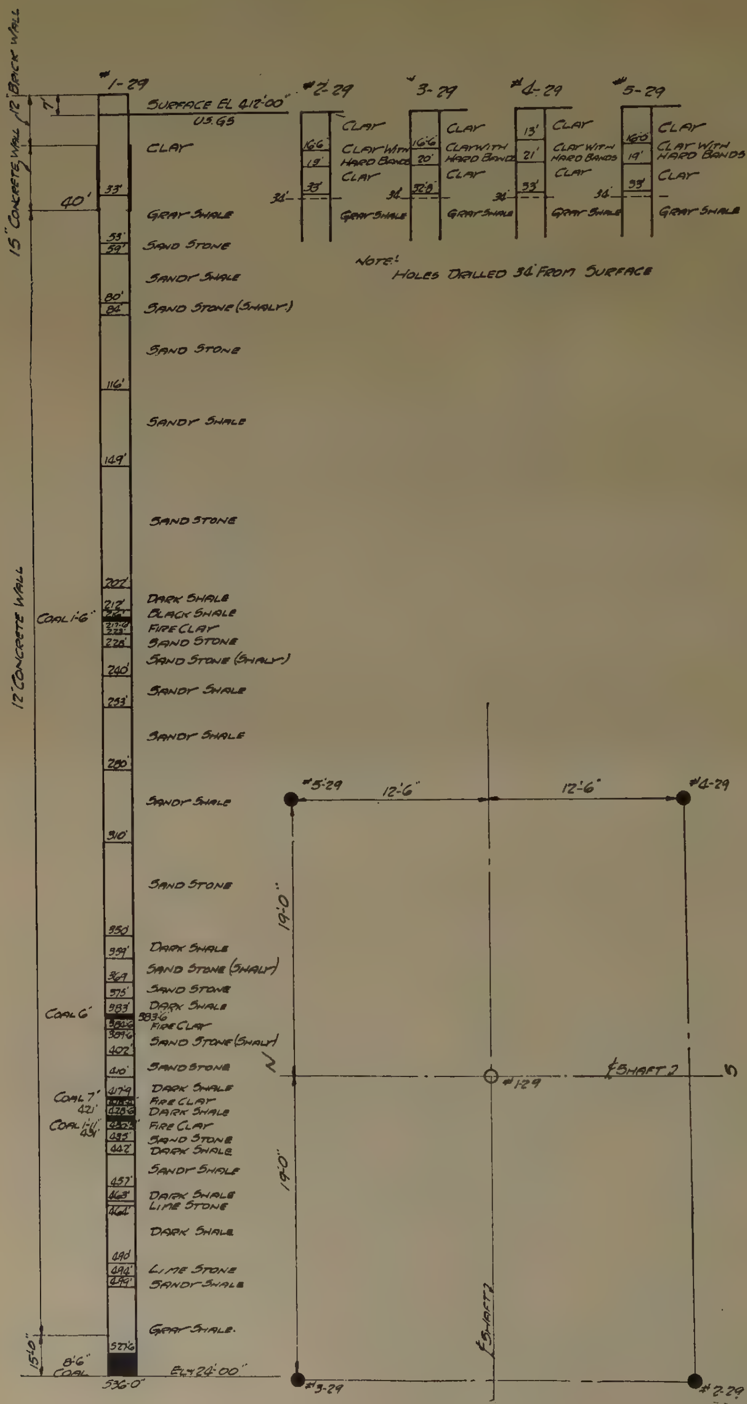


Fig. 5.—Log and location of drill holes for new air shaft, New Orient Mine.
Prospect 1–29, at center of shaft, was finished April 21, 1929.
Prospects 2–29, 3–29, 4–29 and 5–29, outside of proposed shaft, were finished April 24, 1929.

The water gage required to deliver this air was 5 in. These results show a relief from the rapidly increasing resistance caused by the fast-

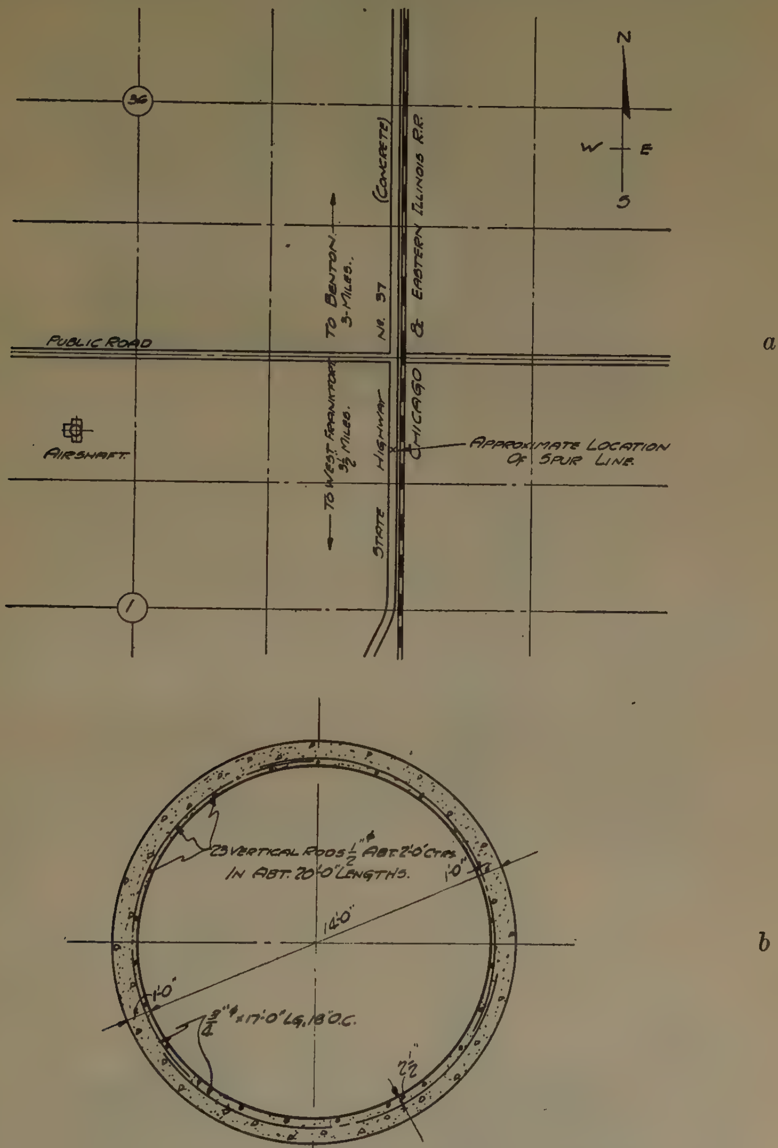


FIG. 6.—LOCATION SKETCH AND TYPICAL SECTION OF NEW AIR SHAFT, NEW ORIENT MINE. Cross-section area of shaft is 154 square feet.

lengthening entries, the volume having decreased from 300,000 cu. ft. per min. at 5-in. water gage at the end of 1928 to 260,000 cu. ft. per min. at 5-in. water gage by the end of 1929. This relief has been tempered,

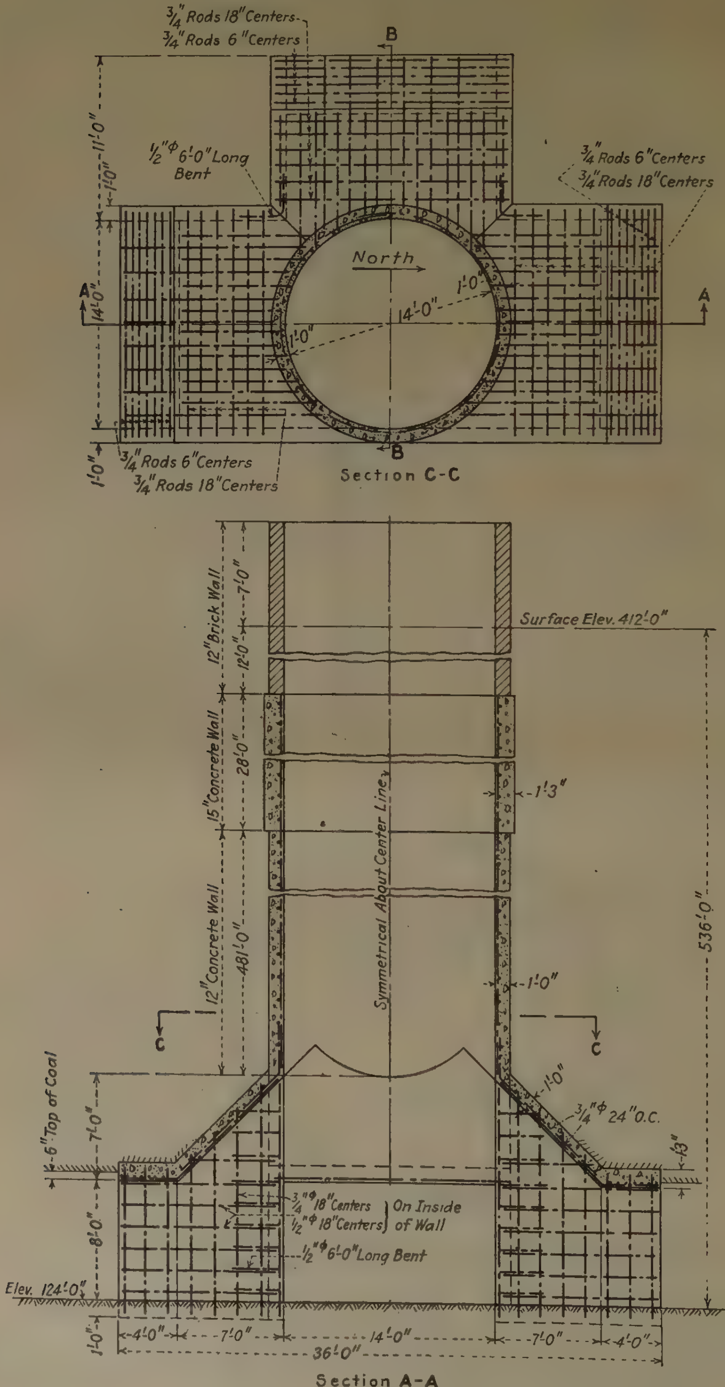


FIG. 7.—SECTION A-A, AND C-C, NEW AIR SHAFT, NEW ORIENT MINE.

however, because only two entries have been connected at present and the resistance is still high. As the workings progress toward the north the volume of return air discharged from the new upcast shaft will increase, and when the second or downcast shaft is sunk, fresh air may easily be maintained on the haulage. It may be advisable, however, to continue the method in effect at present and keep the haulage on the return airways. If so, the air flow would have to be reversed and the new air shaft just completed would be changed to a downcast shaft.

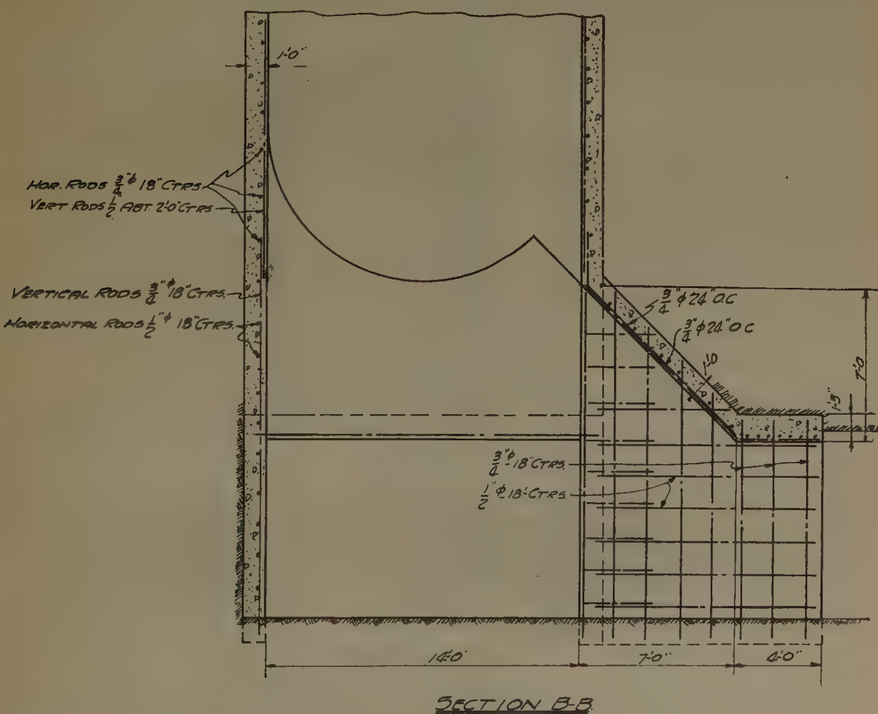


FIG. 8.—SECTION B-B, NEW AIR SHAFT, NEW ORIENT MINE.

The exact location for the second new air shaft, whether it be downcast or upcast, will be determined only after giving full consideration to the flow of air, the number of overcasts required and the ease with which connection can be made to the existing workings.

In order to insure immediate connection between the entries and the shaft as soon as the latter was completed, it was necessary to drive the entries northward on the main north as fast as possible. This was facilitated by the use of two McKinley entry drivers. These machines, driving as much as 1350 ft. per month, or an average of 26 ft. per shift, completed two entries in approximately the same time that was used for shaft sinking.

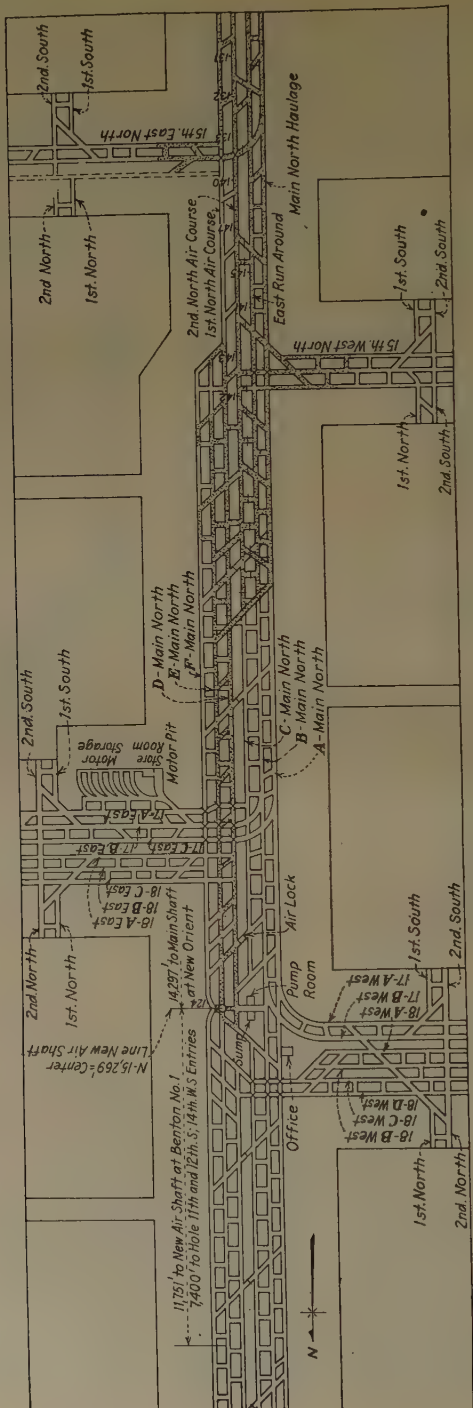


FIG. 9.—LAYOUT OF AIRWAY CONNECTIONS TO NEW AIR SHAFT.

Main Entry Considerations

Referring to Fig. 3, the calculations show that it is possible to continue the four-entry system provided a three-way development is undertaken in the north half of the acreage. However, the water gage is consistently high and there is not much variation possible in the layout of the workings.

On the other hand, by driving all new main headings inbye the present room territory on a six-entry system, it is possible to maintain a low water gage throughout the life of the mine, with the consequent saving in power cost. With this system, it is practical to arrange the workings in a number of ways not possible with four entries. Furthermore, a relatively low water gage can be maintained even if the shaft at Benton No. 1 mine is not available.

Against this system must be considered the loss of a certain amount of room coal tonnage to the additional two entries. If it were assumed that entry coal is mined out at cost (no profit), it is interesting to note that this profit is not entirely lost for, disregarding all other advantages, part of this profit is regained by lower power costs due to the additional entries.

Summary

Summarizing the analysis of conditions at New Orient mine, it was found that with the desired total volume of 400,000 cu. ft. per min., it was advisable to change from the present four-entry system to six-entry headings in all development work, in order to attain the best ventilation conditions and lowest power costs. It was also deemed advantageous to sink a new air shaft immediately to relieve the present conditions and to make the best provision for the future.

DISCUSSION

W. R. Chedsey, State College, Pa., congratulated Mr. Hebley on the paper and said that some other large companies apparently had failed to go into their ventilation problems as thoroughly as the New Orient management had done.

A. C. Callen, Urbana, Ill., asked how the value of K was derived.

Mr. Hebley stated that data taken by the mine engineering staff were used.

E. A. Holbrook, Pittsburgh, Pa., expressed interest in the use of a circular shaft and wondered if many existing rectangular shafts could profitably be converted into circular or elliptical ones. Mr. Hebley was doubtful about this, stating that sinking conditions were easy at New Orient and favorable to the form work required for a circular shaft.

C. Evans, Jr., Scranton, Pa., asked if consideration had been given to skin friction underground.

Mr. Hebley pointed out that the McKinley entry driver gives a favorably shaped and fairly smooth entry. Old entries were cleaned and trimmed up and but little timbering was required.

C. Evans, Jr., asked if horizontal bends within the mine were going to be rounded off like the shaft bottom bends.

Mr. Hebley answered that they are going to use vanes in the bends at the Wildwood mine in Pennsylvania.

R. D. Hall, New York, N. Y., inquired, in this connection, if the floor of the shaft bottom, as well as the roof, should not be rounded off.

Mr. Hebley stated that that is the intention at New Orient.

D. Harrington, Washington, D. C., on inquiry, was informed that the distance between the present and the new shafts is about 3 miles.

Mr. Harrington said that in some work done with three models in a 150-ft. raise at Butte, Mont., one of which was rectangular with timbers, another rectangular but smooth-lined and the third circular and smooth-lined, it was found that the smooth-lined rectangular model delivered about twice as much air as the timbered one, while the circular model gave 3½ times as much.

T. D. Thomas, Lansford, Pa. (written discussion).—Mr. Hebley determined the value for the coefficient of friction (K) for smooth-lined shafts, which he states is in close accordance with other tests as conducted by the U. S. Bureau of Mines and the University of Illinois Engineering Experiment Station. In Bureau of Mines *Bulletin* 261, "Resistance of Metal-mine Airways," by G. W. McElroy and A. S. Richardson, the same subject is very well discussed with a detailed account of all tests made.

The data, as presented, should be of permanent interest to mining engineers, but the value of the coefficient of friction as determined for the New Orient mine is not applicable to all bituminous mines.

On page 15, location of shaft is very important. It would be of interest from an engineering standpoint to know just what the conditions were that determined the location of the shafts in the remote southeast portion of the field.

In Fig. 1, Benton No. 1 mine and Benton No. 2 mine are shown as robbed completely. In case the New Orient mine owners also owned the Benton field, why was this territory abandoned rather than maintain that section so that a connection of the Main North entries could be made, thus utilizing the Benton No. 1 and No. 2 shafts to relieve the high pressure on the air as encountered in 1929? It seems to me that much use might have been made of the Benton No. 1 and No. 2 shafts in ventilating the northern section of the territory by placing an exhaust (primarily) fan on the new shaft near the center of the field.

In the Pennsylvania Mine Code, electric haulage is prohibited on return airways, which in my opinion is a good safety measure. I observe that the haulage way in New Orient is also used as a return airway, which in gaseous mines is not recommended.

Coal-mining Operations in the Sydney Coal Field

By Alex. L. Hay,* Glace Bay, N. S.

(New York Meeting, February, 1929)

The Sydney coal field, the largest and most valuable in Nova Scotia, is situated on the northeastern coast of the Island of Cape Breton, extending from Mira Bay on the south to Cape Dauphin on the north, a distance of 30 miles, and has a general dip northeast under the sea. (Fig. 1.) The field embraces the mining villages of Morien, Birchgrove and Reserve and the towns of Dominion, Glace Bay, Waterford and Sydney Mines. The city of Sydney is situated about midway between the northern and southern extremities of the field and on the western fringe of the productive coal measures.

Sydney harbor crosses the coal field, cutting off direct rail connection between Sydney Mines and the greater part of the field to the southeast. (Fig. 2.) The coal measures are exposed in the cliffs on the coast line, which rise to an average height of 30 ft., and are almost continuous throughout the length of the field.

The portion of the coal field under land forms a small segment of a circle, the greatest distance from the coast landwards is 9 miles, and it occupies an area of 200 sq. miles. The greatest distance developed seawards from the shore by mining operations is 2¼ miles. The strata maintain a regularity of dip at this distance, proving a submarine area of about 70 sq. miles. How much further the coal field extends seawards is unknown, but the indications are that it extends far beyond the present economical working limit and in all probability there is a vast submarine coal field.

The productive coal measures are of Carboniferous age, these being underlain by beds of sandstone, locally classed as millstone grit, which define the southern limit of the field, while a spur of syenitic hills cuts off the coal-bearing strata on the north. Two main anticlinal folds having a general course easterly divide the field into three basins named from north to south: Sydney Mines-Lingan, Glace Bay and Morien.

So far as the portion of the coal field already developed is concerned the general effect of these folds is to vary the direction and dip of the strata at different points of the several basins. The strata dip seawards at the average rate of 6 per cent. along the axes of the basins. On both sides of the several basins, the strata have a dip varying from 7 to 40

* Assistant Mining Engineer, Dominion Coal Co., Limited.

per cent., but generally the rate of the dip in the workings varies from 7 to 22 per cent. The average inclination of the ocean bed is about 2

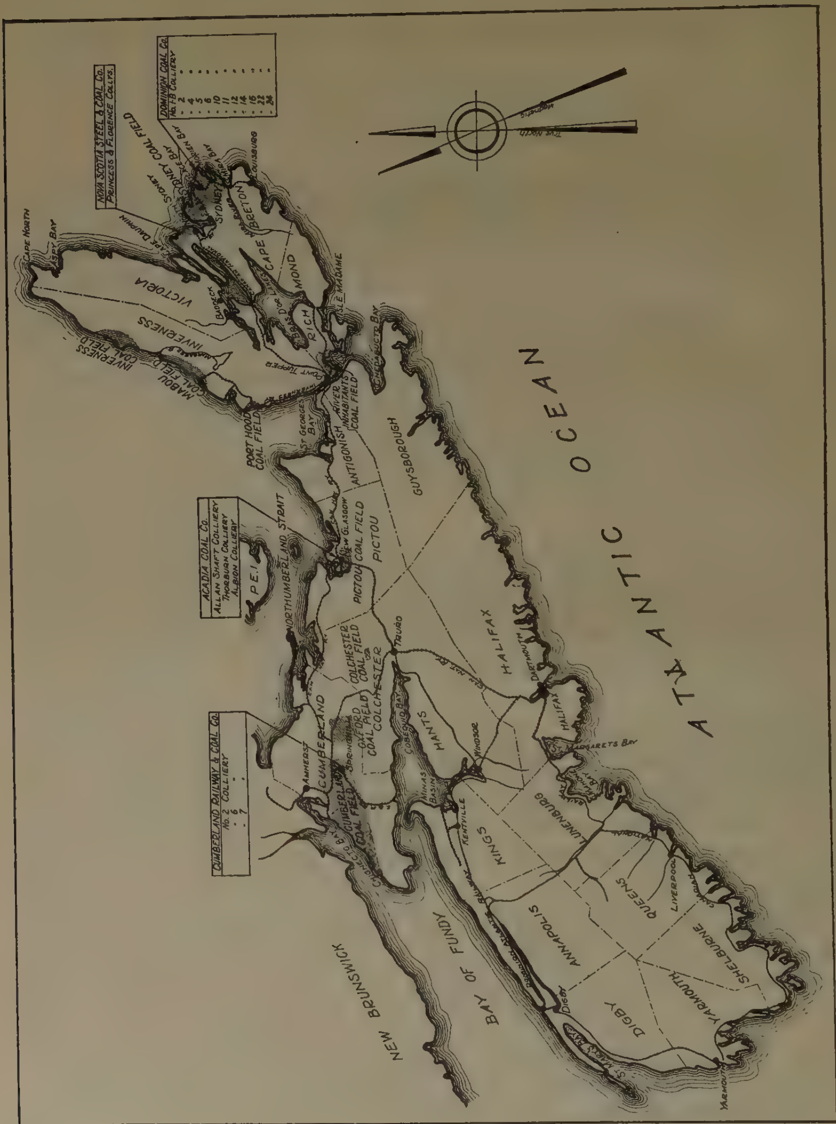


FIG. 1.—PROVINCE OF NOVA SCOTIA SHOWING PRINCIPAL COAL FIELDS.

per cent., so that the overburden of solid measures increases at the rate of 4 per cent.

With one exception all the seams now being worked give off gas, but they cannot be termed highly gaseous.

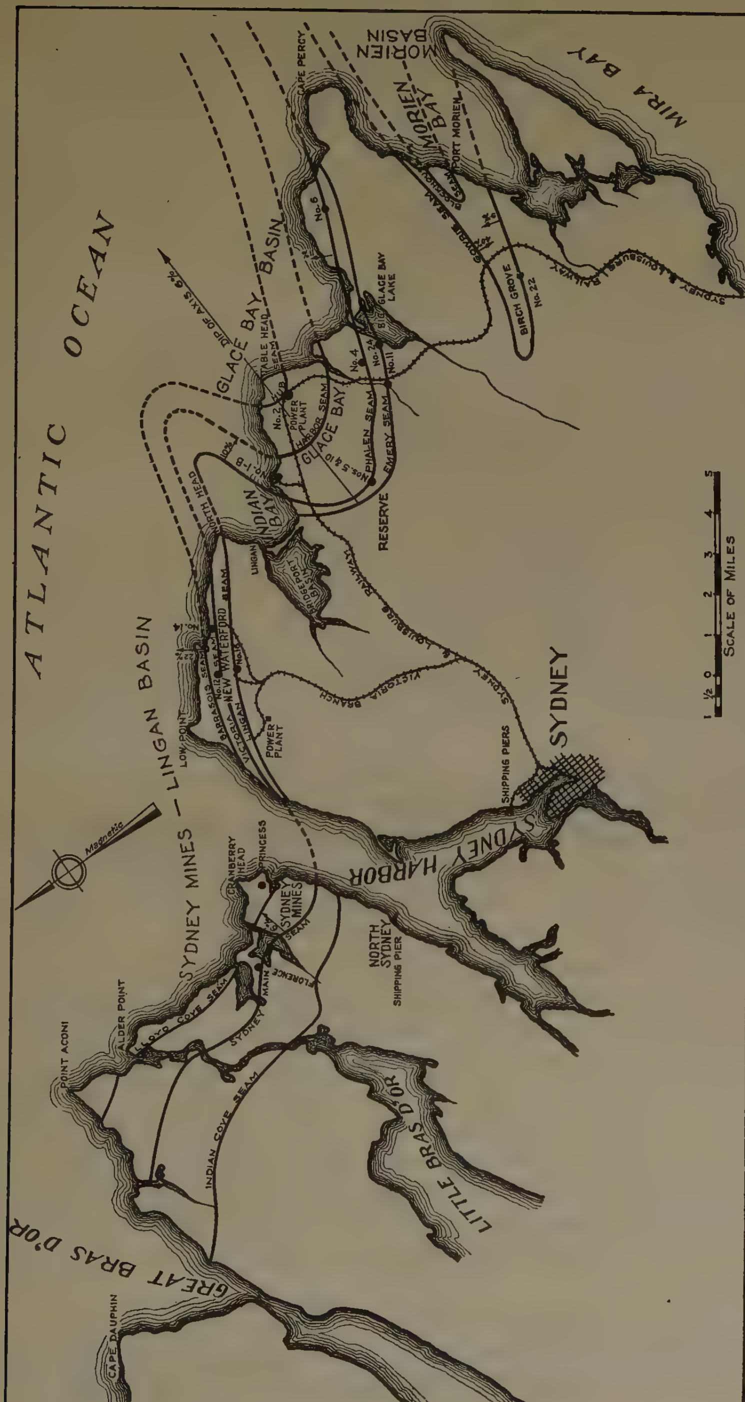


FIG. 2.—SYDNEY COAL FIELD.

The seams in the various basins in the descending order are as follows:

Morien Basin		Glace Bay Basin		Lingan Basin		Sydney Mines Basin	
						Cranberry Hd.	3′ 7″
						Strata—	250′
						Lloyd Cove	3′ 9″
						Strata—	270′
		Hub	4′ 7″	Barrasois	5′ 0″	Chapel Pt.	3′ 9″
		Strata—	375′	Strata—	380′	Strata—	320′
Blockhouse	8′ 0″	Harbor	5′ 8″	Victoria	6′ 6″	Main Seam	4′ 10″
Strata—	570′	Strata—	250′	Strata—	235′	Strata—	430′
Gowrie	5′ 0″	Boutilier	3′ 9″	Fairyhouse	3′ 0″	Indian Cove	3′ 6″
Strata—	210′	Strata—	90′	Strata—	75′	Strata—	215′
Spencer	3′ 6″	Backpit	3′ 0″	Northern Head	4′ 0″	Collins	3′ 0′
Strata—	340′	Strata—	112′	Strata—	75′		
Long Beach	3′ 0″	Phalen	7′ 0″	Lingan	5′ 6″		
Strata—	650′	Strata—	130′	Strata—	900′		
Coal Brook	3′ 6″	Emery	3′ 6″	Mullins	5′ 0″		
Strata—	600′	Strata—	425′				
Tracey	5′ 0″	Gardiner	4′ 3″				
		Strata—	475′				
		Mullins	4′ 6″				
		Strata—	1600′?				
		Tracey	5′ 0″?				

The Blockhouse seam in the Morien basin is correlated with the Harbor, Victoria or Main seam of the Glace Bay and Lingan-Sydney Mines basins, respectively. The continuity of the other seams at workable thicknesses is not as persistent throughout the three basins as is the case of the seam mentioned.

Practically the whole of the known Sydney coal field is under lease to the Dominion Coal Co., Ltd., and Nova Scotia Steel & Coal Co., Ltd., there being only two or three independent companies operating on a small scale in the Sydney Mines district.

The coal is bituminous. It is of a weak and friable nature and because of this presents difficulties in transportation and handling. The seams of quality suitable for metallurgical purposes are the Main or Victoria seam and Lingan seam, in the Lingan-Sydney Mines Basin, also the Harbor, Phalen and Emery in the Glace Bay basin. In the latter basin, the sulfur and ash content increases progressively in all the seams east of the anticline, which separates the Glace Bay and Lingan-Sydney Mines basins, about 30 per cent. of the proved area in this basin being suitable for metallurgical purposes. The coals of the seams worked at present coke readily, the Phalen seam giving the best results in this respect.

The roof overlying the working areas is for the most part a weak shale, its weakness becoming more evident as the mines are developed seawards under greater cover. The roof over the Gardiner and Emery

seams, in limited areas of the Glace Bay basin, consists of sandstone. The floor ranges from soft fireclay to hard shale. The strata of the productive measures in the Glace Bay basin consist of the following: Coal, 2 per cent.; shales, 60 per cent.; fireclay, 15 per cent.; sandstone, 23 per cent. The field is remarkably free from strata dislocations or "wants," although in the vicinity of the anticlines there are irregular dips and rolls.

History

Coal was first mined in the Sydney coal field by the French military authorities in the year 1720, an opening being made on the Blockhouse seam in the Morien basin. For the next 100 years crop mining was carried on in a small way in various parts of the field. In the year 1825, the General Mining Association was formed and, two years later, an effort made towards systematic mining in the Sydney Mines-Lingan basin. The Crown in 1849 conveyed its interest in the minerals of the Province to the Government of Nova Scotia. In 1857, the General Mining Association which till then had held under sublease all the mineral areas of the Province, surrendered, in consideration of certain concessions and privileges, its holdings to the Government of Nova Scotia, reserving, however, for its own operation certain areas in Cumberland, Pictou and Cape Breton counties, totaling 30 sq. miles. Shortly afterwards several mining companies were formed and mines were opened up in the Glace Bay district. In the year 1893 seven of the then eight operating companies were consolidated to form the Dominion Coal Co., Ltd. The General Mining Association, however, continued independent operations at Sydney Mines. In 1901 this company sold out to the Nova Scotia Steel & Coal Co., the latter merging with the Dominion Coal Co., and others of allied interests, to form the British Empire Steel Corpn., Ltd., in 1921.

The output from the Sydney coal field approximates 21,000 tons per day. At the present time the Sydney Mines-Lingan basin contributes 35 per cent., the Morien basin 4 per cent. of the total output. The balance of the output—61 per cent.—is mined from the Glace Bay basin.

Owing to submarine conditions it is not possible to determine accurately the tonnage recoverable from this field. Already approximately 140,000,000 tons have been extracted and it is estimated that the tonnage of economically recoverable coal still available within proved areas, and of a quality equal to that being mined at the present time, exceeds 1,000,000,000 tons, and that the recoverable coal within the limits of five miles seawards from the shore is approximately 2,500,000,000 tons, or sufficient to last for 600 years at the present rate of production,

Phalen Seam

The Phalen seam, which is 7 ft. in height, is the most important one in the field and has been extensively worked. The land area of this seam—15 sq. miles—is practically exhausted. The submarine area worked over to date is 12 sq. miles. Assuming a working limit 3 miles seawards from the shore, there remains over 30 sq. miles yet to be mined. There are six operating collieries on the seam, Nos. 1B, 2, 4, 5, 16 and 22. All these collieries are shown on the plan, except No. 22, which is in the Morien basin.

No. 5 colliery has been operating since 1872 and has produced 12,500,000 tons since that year. It is nearing exhaustion now. The other mines have a combined daily output of 11,000 tons. The oldest colliery in the field and also the latest mine opened are both operating on this seam, the former No. 4 or Caledonia and the latter No. 1B.

Caledonia mine was opened in 1866 and has worked continuously since. Output records prior to 1894 are not available, however it is estimated that 2,500,000 tons were extracted previous to that year. Since then the mine has produced 13,170,000 long tons. Its present daily production is 2000 tons and it is estimated that another 10,000,000 tons will be mined from the Phalen seam at this colliery. The maximum cover under which coal is being extracted is 1300 ft. The greatest distance from the shaft bottom to the working face is 3.6 miles. Men are transported by means of riding rakes a distance of 2 miles. The main haulage, which is of the endless type, is 3 miles long, 2 miles of this being under the sea.

Dominion No. 1B Colliery

No. 1B Colliery, as already mentioned, is the latest colliery in the field. It was opened in June, 1924, and was constructed and equipped to win a large area of submarine coal which was made available when the Nova Scotia Steel & Coal Co. merged with the Dominion Coal Co., forming a part of the merger known as the British Empire Steel Corpn. The following description is culled from a paper prepared by the writer two years ago, for the Engineering Institute of Canada:

In 1893 a shaft was sunk at Dominion No. 1. Seven years later another shaft was sunk at what is now known as Dominion No. 2. The object of these two shafts was to afford entrance to win all the coal held under lease by the Dominion Coal Co. in this section of the field. All coal to the dip of the barrier was allocated to Dominion No. 2 and the remainder to Dominion No. 1.

In 1914, considerable difficulty began to be experienced in maintaining satisfactory ventilation. This condition had developed gradually, but as the airways became lengthened, the difficulty was accentuated.

Increasing the ventilating pressure would have resulted in very heavy leakage losses, without improving the ventilation to any appreciable extent. The cost of opening up these airways and building concrete stoppings was considered prohibitive.

The distance a miner was compelled to walk from the shaft to his working face was in some cases upwards of 2 miles. When it is remembered that a miner's work is usually strenuous, it can be seen that the time and energy expended in walking 4 or 5 miles a day underground was unprofitable to both the miner and the company. The roadways in years gone by had not always been developed with the end in view of ultimately making them haulageways for the transportation of men, with the result that a very heavy expenditure would have had to be made if these roadways were to be built at this late date.

Another point that was being given serious study at this time was the desirability of establishing an additional escapeway from No. 2 colliery against the possibility of a disaster, such as a fire or explosion. At that time there were three entries into No. 2 mine. One of these entries was via No. 5 colliery. As this latter colliery was gradually reaching the point of exhaustion, it was deemed advisable to provide another escapeway.

With these objects in view—improving the ventilation of No. 1 mine, reducing to a minimum the length of walk for the miner of No. 1 colliery, and providing an additional escapeway for the miners of No. 2—it was decided to sink a shaft to No. 1 mine as close to the working face as possible and build a branch line from the Sydney & Louisburg Railway for the transportation of men on the surface.

It will be seen from Fig. 3 that the seam had been worked out entirely under the land and also for a very considerable area under the sea. The narrow strip which formed the boundary between Dominion Nos. 1 and 2 collieries still remained intact and afforded the only suitable site for a shaft. This explains why the shafts are located within 55 yd. of the sea cliff, which has an average erosion of 8 in. a year. Protection against erosion is afforded by dumping refuse from the pits along the cliff side, a railway track being laid parallel to the cliff for this purpose.

A circular shaft was sunk and completed as a man shaft and return airway in 1921. The shaft was 670 ft. deep and 12 ft. dia. It was lined with brick for a distance of 213 ft. from the surface and below this point was lined with concrete. For a time the shaft was wet, owing to seepage of sea water. Holes were therefore drilled through the concrete lining and cement grout pumped in under pressure. This practically eliminated the leakage and the shaft today is dry.

The long haul necessary to win the coal from the outward limits of the Dominion Coal Co.'s area to Dominion No. 1 shaft was adding very considerably to production costs. On the other hand, the limited ton-

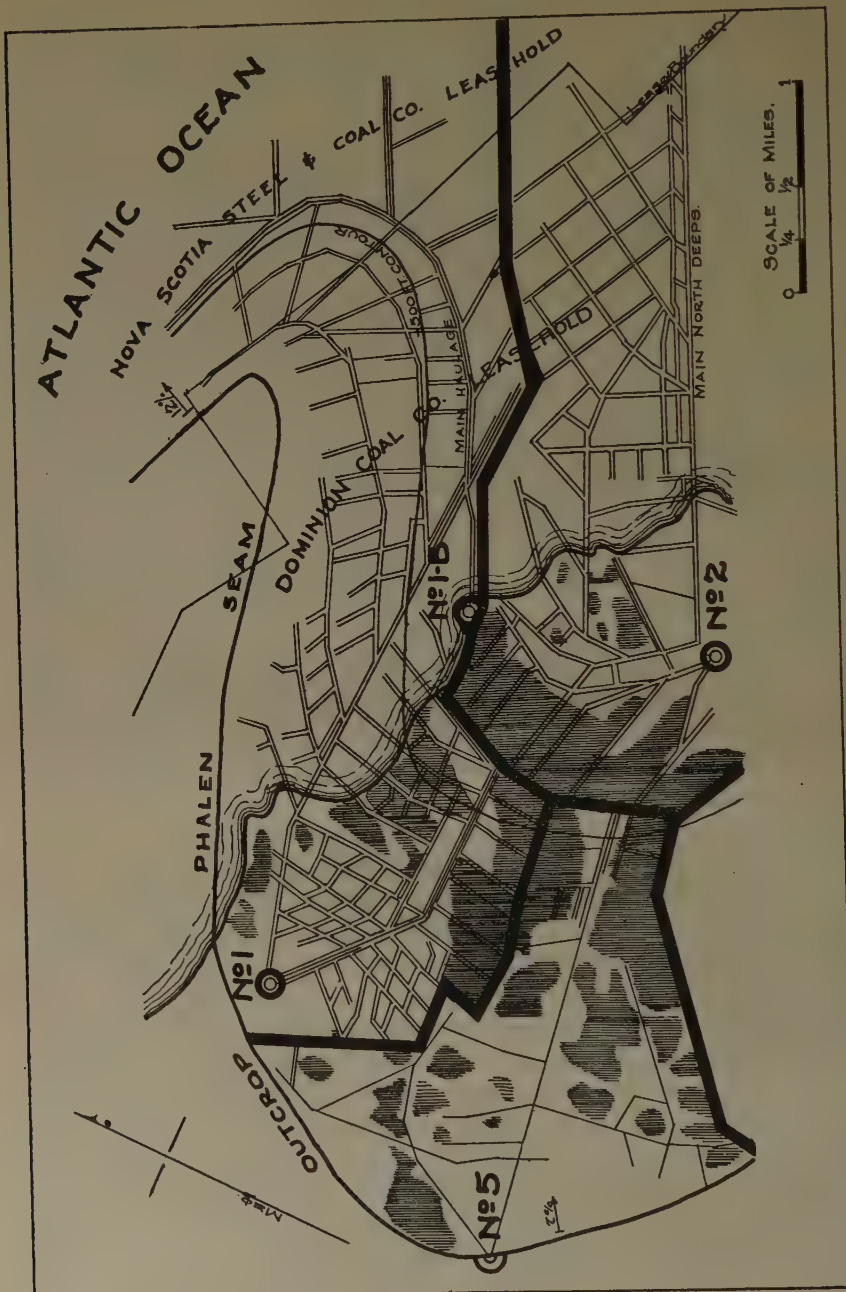


FIG. 3.—NOS. 1, 1B AND 2 COLLIERIES.

nage available from this area would not justify the cost of sinking a coal shaft and erecting a new surface plant. The economical mining limit for this mine was being approached rapidly when the merger resulting in the British Empire Steel Corpn., Ltd., was consummated. This completely changed the aspect of things, since it gave access to a large area of coal held under lease by the Nova Scotia Steel & Coal Co., seawards of the Dominion Coal Co.'s leasehold (see Fig. 3). Under these circumstances it was considered advisable to sink a coal shaft near the air shaft, which was the only possible site. The advantages of such a decision are obvious: the length of haul would be reduced nearly 2 miles, maintenance of compressed air line and main airways would be reduced by a similar distance, a modern and more economical screening and surface plant could be erected and production could be doubled.

Coal Shaft

The coal shaft was located on July 30, 1921, and sinking operations commenced the following month. As already stated the location was restricted to certain narrow limits, which hampered the layout to some extent both on the surface and underground, but particularly underground. Since the life of this colliery is estimated at upwards of 100 years, it was judged to be true economy to give every structure the character of permanence and this was the dominant note throughout.

The shaft is 670 ft. deep and 31 ft. 2 in. by 13 ft. 4 in. in the clear. The shaft collar is of concrete and extends 32 ft. from the surface. Below this point the shaft is timbered with 8 by 10-in. buntons and wall plates, and 6 by 10-in. posts, placed at 6-ft. intervals, sheathed skin-tight, with 3-in. pine lagging. At 36-ft. intervals an additional set of timbers was put in, hitched into the shaft wall for additional support. The shaft timbers are mortised in such a manner that all are interlocked, obviating the use of nails or spikes. All the timber used in the shaft has been treated with creosote. A shaft ring about 230 ft. from the surface drains the water off the shaft into a pipe and the shaft is comparatively dry.

There are three compartments to the shaft. Two of these, each 11 ft. 3½ in. by 12 ft., are used for hoisting coal and are fitted with double steel guides of 85-lb. rail section The third compartment, 6 ft. 3 in. by 12 ft., is used as a pipeway. At the present time there is one 12-in. air line, one 14-in. cast iron water discharge column and a 3-in. water-supply pipe in this compartment.

There is a rock tunnel, 5 by 8 ft., driven from the north end of the shaft at a point 34 ft. below the surface to the face of the cliff. The mine water, 1250 g.p.m., is discharged through the 14-in. shaft column to this drift, whence it flows into the sea at a point 9 ft. above mean sea level.

At the west side of the shaft and at a point 112 ft. above the shaft bottom an opening 8 by 15 ft. was made into the Back Pit seam. Connection is made through a tunnel having an inclination of 55° with the workings of No. 1B. This serves as a return airway for the colliery.

Shaft Bottom

At the site of the shaft, workings had already been developed without regard to the requirements of a new mine. Consequently in the layout of the pit bottom, the roads in existence had to be utilized whether of

FIG. 4.—APPROACH TO SHAFT, NO. 2 COLLIERY.

suitable grade or not. This necessitated very heavy rock grading. The amount of rock blasted to meet the requirements of standage exceeded 8000 cu. yd. The pillars in the vicinity of the shaft were very small and the rock blasted from the roadways for height and grade was used to fill the open spaces, thus reenforcing the coal pillars and arresting the process of coal pillar disintegration.

Due to the fact that the shaft was sunk close to the workings of No. 2 colliery, where all the coal had been extracted, there was a possibility of a movement of the overlying strata in the vicinity of the shaft. Any such movement would prove disastrous to the colliery, consequently special precautions were taken to prevent this. For this reason the approach to the shaft, for a distance of 48 ft. on either side, was secured by fabricated steel sets, roof, floor and side members being tied together. The roof members vary from 17 to 23 ft. in length and are 18 in. in

depth, with flanges of 11½ in. The floor members are 12-in. I-beams, with 5½-in. flanges. The vertical members are 8-in. I-beams, except in the immediate vicinity of the shaft, where 10-in. H-beams are used. (See Fig. 4.) Concrete walls were erected along the sides of the opening, into which the vertical steel supports were bedded and concrete filling arched between the roof members. To give further support concrete walls were carried for a distance of 600 ft. from the shaft. These walls vary from 12 to 15 in. in thickness, depending on the roof members they carry and on the condition of the pillars contiguous to them. The roof throughout the roadways of the pit bottom standage is supported by 10-in. steel H-beams of Bethelehem section. The walls of the empty and full roads inbye the point where the concrete walls were built were lined with rock from the excavation of the road and the nearby ventilation tunnel, and were given a face of gunite. The main objects here were the preservation of the coal pillars and the prevention of the forming of coal dust from the crumbling of the ribs, the coal itself being quite friable.

Movement of Output

By referring to Fig. 5, the movement of the output may be readily understood. Trips of 50 or more cars, of 2 tons capacity, are hauled by

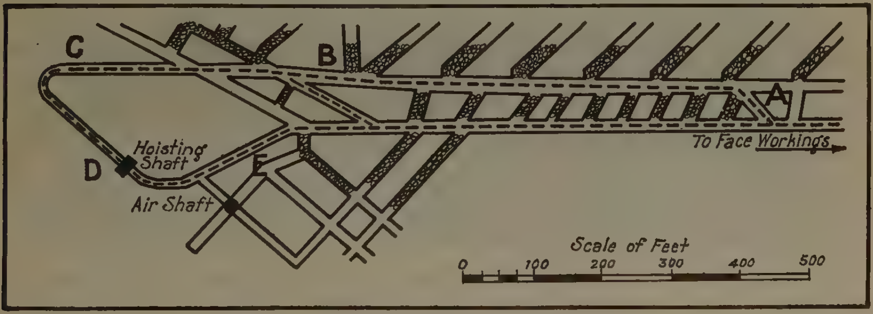


Fig. 5.—Plan of No. 1B pit bottom.

trolley locomotives through the main turnout A to the standage AB. The gradient of this standage is 9⁄10 per cent., favoring the load. The locomotive uncouples at B and runs through the runaround track.

The trip is spragged and moves slowly to point C, where a motor-driven creeper chain engages the car axle and hauls the car to an elevation which permits a gravity run to the shaft bottom. One man is employed at B to uncouple the cars and operate the chain. At C there is an automatic switching device, which operates to throw the switch points after the passage of two cars. At D two automatic switches throw the points

alternately. In this way cars are fed to the shaft bottom on four tracks.

Near the shaft the cars are held by a car stop, which is operated by the seating of the descending coal cage. As soon as the cage comes to rest, stops in the cage itself are released and at the same time the car chocks in the full road are thrown open. The cars next to the cage, brought to rest on a 3 per cent. grade, move forward by gravity and bump the empties off the cage. The entry of the full cars throws the pit bottom and cage chocks into service. This automatic caging arrangement is known as the Nolan cager. Two cars are caged side by side simultaneously and similarly two empties are pushed out on the empty tracks.

By means of a very simple automatic chocking device the cars are regulated to a single track. One car has a clear track to the empty standage, while the second car is retarded by a chock 8½ ft. from the shaft. When the first car has travelled 30 ft., the wheel of the car engages with the lever arm of the retarder and releases the second car (Fig. 4).

The cars run by gravity to point *E*. At this point a creeper chain, similar to one on the full road, engages the car axle and hauls the car to an elevation sufficient to deliver it to the level empty standage. The cars are coupled at this point and a motor-driven pusher pushes the trip forward. The locomotive picks up the trip at *A* and hauls it to the inside workings.

The question of skip hoisting was investigated but because more than one seam of varying qualities will eventually be worked concurrently, it was deemed advisable to adopt a system of hoisting whereby coals from the different seams might be kept separate.

There is a locomotive repair shop close to the pit bottom which has two stalls with pits and is equipped with a crane. Here repairs can be readily effected.

The grades throughout, in the vicinity of the pit bottom, are designed to take care of a constant stream of cars, that is to say, between the full and empty creeper chains the grades are sufficiently great to permit a box to start readily from rest and move forward its own length only and this without excessive jar or shock. If there is any delay in delivering coal from the mine to the pit bottom and hoisting is continued until all cars are removed between the two creeper chains, it is evident that cars released at the full creeper chain knuckle will acquire a very considerable momentum before reaching the shaft.

As designed, three men are sufficient to handle 2500 tons in 8 hr., that is, one man uncoupling and running the full road creeper, the motor for this having a remote control, one man signalling the shaft hoist and one man coupling the empties and operating the empty creeper.

Pumping

To centralize the pumping of Nos. 1 and 5 collieries, the combined water growth of which is 1250 g.p.m., it was decided to impound the water in one main lodgment near No. 1B shaft, advantage being taken of existing barriers for this purpose. The make of water at No. 1B itself, being less than 60 g.p.m., is allowed to drain through 2¾-in. boreholes driven through the barrier to No. 2 mine. This latter mine is naturally dry, and only part of the water entering through these boreholes has to be pumped as the ventilating current takes up a considerable portion of it.

The pump room is located in an old chamber which was later excavated to a height of 14 ft. and the ribs reenforced with 12-in. concrete

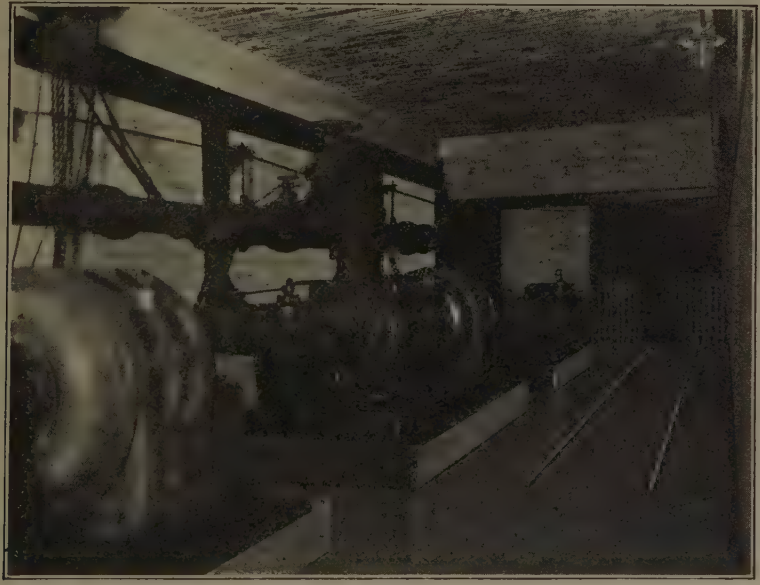


Fig. 6. Pump room.

walls. The roof is supported with 85-lb. rails, set at 9-in. centers and lagged with brick (Fig. 6).

The pumping plant has two three-stage Sulzer centrifugal pumps, each direct-connected to a 400-hp., 2200-volt motor, running at 1500 r.p.m. Each pump has a delivery of 1250 g.p.m., against a head of 630 ft. There is a third pump of the reciprocating type, motor-driven, which pumps the water from the shaft sump into the suction of the Sulzer pumps.

An ell of the pump room contains step-down transformers for the lighting of the pit bottom and the operation of the car haul motors. A split of fresh air, conducted through a 12-in. wrought iron pipe from the main air course, ventilates the pump room.

Development

As already mentioned, a very large area of coal is allocated to this mine. It is difficult to say just where the economic limiting line exists, but it is reasonable to assume that as the years go by and advances are made in engineering practice, this limit will be extended farther seaward.

By referring to Fig. 3, it will be seen that the contour of the seam swings through an arc of 135°, as it turns around the anticline separating the Glace Bay and Lingan basins. While the shaft is located in the Glace Bay basin, it is apparent that the bulk of the coal to be drawn to No. 1B shaft will be from the Lingan basin. The seam is 7 ft. thick, clean coal, and is suitable for metallurgical purposes.

By means of cross-measures drifts from the Phalen seam, overlying and underlying seams will eventually be won. The tonnage of coal recoverable from these seams from a boundary line considered within the present economical limit is: Hub, 18,000,000 tons; Harbour, 27,000,000 tons; Phalen, 43,000,000 tons; Emery, 22,000,000 tons; Gardiner, 30,000,000 tons, or a total of 140,000,000 tons. With an output of 2500 tons per day and continuous operation, *i. e.*, 280 days per year, the Phalen and upper seams alone will make the life of the colliery 126 years. If the lower seams are added, the life of the colliery will be 200 years.

The Phalen seam only is being worked at the present time. The main development to date is a haulageway which follows approximately the contour of the seam from the shaft bottom. At the time of writing the face of this roadway is 3.3 miles from the shaft. It will continue to drive along the contour until it reaches the western barrier, 4.3 miles distant from the shaft. This barrier is the eastern limit of an area reserved for development from North Head, Lingan.

At a point 1 mile from the barrier it is proposed to develop deeps on the full dip of the seam for a distance of at least 2 miles. The length of face between the barrier of the reserved area and No. 2 Colliery is 3 miles, and as development is made seawards this will increase, the divergent angle being 27°. As this was considered too great a length of face tributary to one deep, another set of deeps was driven from the main locomotive haulage, 10,600 ft. distant from the shaft, and parallel to No. 2 barrier. This will take care of 1 mile of producing face. By the early winning of these deeps, the output of the mine can be maintained easily while the long development of the farthest deeps is prosecuted.

System of Mining

The system of mining is pillar and room. This is the only system feasible within present cover limits, where 56 per cent. of the coal is left to support the sea bottom. As the cover increases to the dip a point will be reached where all the coal may be safely extracted. This

may be done by removing the pillars formed by first operation or by longwall. Hydraulic stowing has been suggested for this seam, replacing the coal by sand, crushed slag and other refuse. This, however, is not considered economically feasible at the present time.

The coal is cut by air-driven Ingersoll-Rand radial coal cutter machines. These machines are comparatively light, weighing 391 lb., and are easily moved. They are capable of undercutting 75 tons or more per machine per day, although the average perhaps does not exceed 45 tons.

Transportation

The coal is hand-loaded at the working face into cars of 2 tons capacity, of which the principal dimensions are: height above rail, 3 ft. 8 in.; inside length, 8 ft. 8 in.; between buffers, 10 ft. 3 in.; width, 4 ft. 4¾ in. The rail gage is 36 in., wheels are 16-in. tread dia. and are equipped with Timken roller bearings. The cars are built with solid ends and require a complete revolution to empty them. As a result they are comparatively dustproof. The cars are hauled by horses from the producing face to room landings, an average distance of 1300 feet.

Trips of 17 cars are made up on the landings and lifted by a 15 by 16-in. Ingersoll-Rand air hoist and lowered through a headway to the main landing. Three headway trips, or 51 cars, constitute a normal main haulage trip. Two headways are in operation at the present time, the average output from each being 900 tons.

Each air hoist is so designed that it is a combined hoisting and compressing engine. When hoisting, it offers all the advantages of an efficient hoist. When lowering, the design permits of air being compressed into the main pipe line with good efficiency.

The rope speed is controlled by the amount of air compressed and is at all times under control of the operator. By the movement of a lever, more or less air is compressed, thus governing the speed of the rope thereby effecting a direct saving in energy that is usually absorbed by friction brakes.

Operation from hoisting to lowering or compressing is automatic. The hoist is equipped with friction brakes for emergency. A compressor discharge valve is placed in a by-pass to insure automatic passage of the compressed air and to prevent the return of the compressed air when lowering or with the throttle valve open.

Coal from the dips is hauled in 13-car trips from room landings by means of main and tail rope motor-driven hoists. There are two of these hoists in operation, each driven by a 150-hp. motor, at 2200 volts. The coal is hauled from the main landings to the shaft bottom by three 16-ton, 240-hp., d.c., Goodman trolley locomotives. The line pressure is 250 to 275 volts. The locomotives have a drawbar pull of 6500 lb.

on level grade, at a speed of 8 miles per hour. The wheels, 33 in. dia., have cast iron centers with steel tires 2¾ in. thick, the tread being 3 in. wide, and the wheel base 66 in. Each locomotive has two 120-hp. geared motors, centrally hung, of the open type. There is an open-drum controller at each end of the locomotive.

Three of these locomotives provide ample margin in handling the total output from the present workings. It is estimated that the main haulage can be extended another 5000 ft. without additional locomotives.

The system of main haulage at most collieries, both here and on the mainland, is plane rope, endless rope, or a modification of these. It is true that Jubilee colliery, Sydney Mines, had some years ago introduced storage battery locomotives for the main haul which in this case was very short. Also about 20 years ago compressed air locomotives were introduced in Nos. 2 and 9 collieries to meet a condition somewhat similar to that existing in No. 1B today. While these locomotives did the work, they were far from economical and in time were discarded.

About 25 years ago, a trolley locomotive was used in No. 1 mine for comparatively short level hauls. Whether or not it was considered economical in those days, it is certain that it was operated without regard to the possible dangers arising from a dusty road.

At No. 1B, owing to the changing direction of the contour, it was not considered feasible to work this mine with rope haulage with any degree of success. To meet this curvature condition it was necessary to install locomotive haulage of some kind.

The merits of storage battery and trolley locomotives were discussed fully. It was felt that for long runs, such as will be the case in this colliery, trolley locomotives would be the more economical. This decision led to special precautions being taken to prevent dust or gas ignition. It also necessitated the widening and increasing of the height of the roadway. The roadway as originally developed was 7 by 10 ft. It was necessary to increase this to 8 by 20 ft., giving it a finished clearance of 6 ft. 6 in. by 18 ft. This clearance holds good, except for a distance of 1000 yd. between the shaft turnout and the assembly yard, which is laid with single track. Upwards of 14,000 tons of coal and 16,000 cu. yd. of rock had to be excavated in preparing this roadway. The average grade of the main road is 0.5 per cent. rising from the shaft; however, there is one section, 700 ft. long, where the grade is 0.5 per cent. against the load.

The coal, of course, was loaded out to the surface, where it had a market value. The disposal of the stone, however, was a more difficult problem. All openings adjoining the roadway itself were stowed with excavated rock. Although there were numerous vacant chambers to the rise of the parallel roadway, these could only be used to a limited extent, as this latter place was the chief coal road of No. 1 colliery and

intensive development work was being carried on at the same time as construction.

Stone packwalls were built along both ribs of the haulageway. These walls varied in thickness from 1 to 4 ft., depending on the road alignment. This absorbed about 6000 cu. yd. of excavated material. Where the rock was of a sandy nature it was crushed to a size suitable for road ballast. Apart from a small section at the pit bottom, all excavated material was disposed of as outlined above.

The packwalls were given a coat of gunite, 1 cement, 3½ sand. This bonded the walls, closed the crevices and reduced to a minimum irregu-

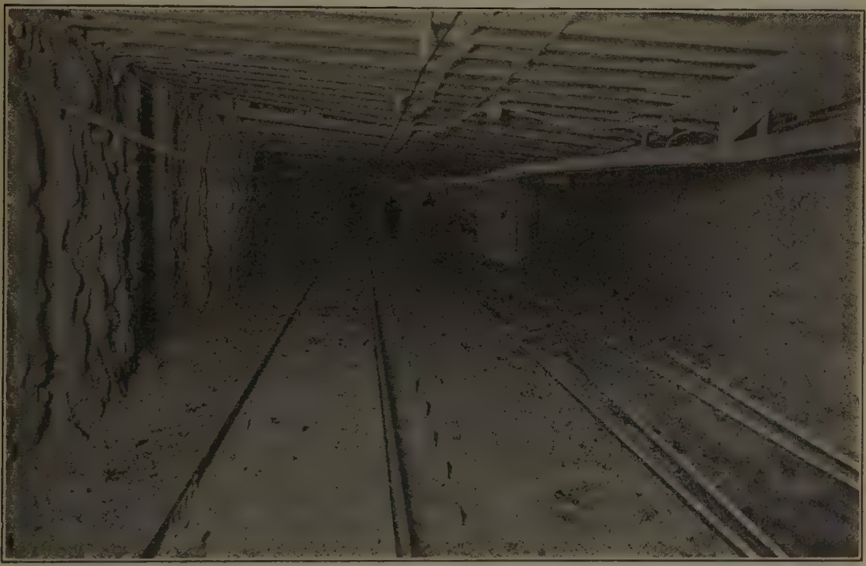


Fig. 7.—Stone packwalls in haulageway, No. 1B colliery.

larities where coal dust might lodge. The walls prevented spalling of the coal ribs, which would have created a very serious dust menace. (Fig. 7.)

The roof, which consists for the most part of soft shale, is supported by rail booms, 60 lb. to the yard where the span does not exceed 12 ft., and 85 and 100-lb. rails where the span exceeds this. Considerable lofting is necesary over the steel booms as the roof deteriorates due to the changing temperature.

In the extension on the mail haulage, now under construction, straight concrete walls 8 in. thick are erected on either side. These support the rail booms. Skin tight lagging is placed between the rails. (Fig. 8.)

The track rails are 60 lb. to the yard, laid on ties 6 by 6 in. by 5 ft., spaced at 30-in. centers. This makes a substantial track for the main line, which must serve for many years. The rails are bonded with copper

bonds at each rail joint and cross-bonded at 200-ft. intervals. At intervals of 2500 ft., the track rail is grounded to the main air pipe on the parallel road. The tracks are laid at 8-ft. centers and as the locomotives are 5 ft., 2 in. in width, this gives a clearance of 2 ft., 10 in. between passing locomotives. This is ample to permit center supports if these should become necessary. There is a clearance of 1 ft. 11 in. between the locomotive and packwall on the low side and 2 ft. 11 in. on the high side. The high side is used by workmen and officials whose duties take them on the haulageway during operations. In addition to the clear space of 2 ft. 11 in., there are manholes established at 100-ft. intervals as a further safety measure.

FIG. 8.—CONCRETE WALLS IN EXTENSION OF MAIN HAULAGE.

The trolley wire is hard drawn copper, 4-0, B. & S. gage, fastened to insulators, secured to the roof beams. It is offset 6 in. east from the low side rail and supported at a height of 6 ft. 4 in. above it. There is a feeder cable carried close to the packwall on the low side of the roadway. The drop at the end of the line does not exceed 45 volts.

Considerable sparking was experienced when operations commenced. This has been very much reduced by fixing the trolley wheel so that it makes a sliding contact.

The 300-kw. C. G. E. motor generator set, which supplies direct current to the trolley system, is located 4600 ft. from the shaft bottom. A 4-0, three-conductor, paper-insulated, lead-covered, armored cable, 2200 volt, is carried from the air shaft along the intake airway to the

motor generator set. The generator is driven by a synchronous motor at 750 r.p.m.

Throughout the entire length of the roadway, stone dust has been used freely to neutralize as much as possible the combustible coal dust. Coal spillage is cleaned regularly from the roadbed. In spite of this, quantities of small coal accumulate and are ground by traffic into dust. Coal dust is also swept off the top of full cars in motion. The combined velocity of the inbye air and the outward bound car creates a very strong current and at places produces a cloud of dust to the rear of the moving trips. Water sprays recently established in landings spray the top of each loaded car before entering the main haulage. This has helped to reduce the coal dust materially. The roadway is examined daily and samples of dust taken at frequent intervals. The degree of fineness is determined and the combustible matter present ascertained. It is aimed to keep the combustible dust below 25 per cent., although as high a percentage as 45 may be considered safe. To prevent the propagation of an explosion, stone dust barriers are erected at intervals of 500 ft. These consist of a number of shelves ladened with dust and easily tripped when subject to concussion. Each battery or barrier is 50 ft. in length and carries 2 tons of stone dust.

The stone dust is derived from dolomite tailings obtained from a quarry at Point Edward near Sydney. A Bradley Junior Griffin mill located in the colliery yard, having a capacity of 1½ to 2 tons per hr., pulverizes the stone so that 80 per cent. passes a 200 mesh. Considerable difficulty was experienced at first owing to the wet condition of the stone supplied to the mill. This prevented successful operation and reduced the capacity of the mill. Later a pre-dryer of the rotary type was installed, which obviated this difficulty.

The stone is dumped from railway cars into a hopper below the level of the railway tracks. From here it is elevated by means of a continuous bucket conveyor to a Canadian Rand jaw crusher, where it is crushed to ½ in. ring and under and then conveyed by a bucket elevator to the rotary dryer. The dryer consists of a steel drum 2 ft. 9 in. dia. and 18 ft. long, housed over a fire grate in a brick chamber, 18 ft. long, 3 ft. 6 in. wide and 5 ft. high, and is thus brought into direct contact with flame, the hot gases of combustion returning through the drum to the stack. The drum makes 10 r.p.m., and since the outlet is 9 in. lower than the intake, there is a continuous flow through the drum, the interval of travel being sufficiently long to dry out all but a small percentage of inherent moisture.

Ventilation

The mine is ventilated by a 5 by 10-ft. Sirocco forcing fan, connected by a Morse silent chain drive to a 400-hp., C. G. E. synchronous motor.

The fan is designed to deliver 300,000 cu. ft. of air per min. at 6 in. w.g. At the present time the fan delivers 90,000 cu. ft. of air per min. with a w.g. of 3.8 in., running at 183 r.p.m. There is a stand-by, 20 by 42 in., Dixon steam engine, connected to the fan shaft by a rope drive. The air is delivered through a staple pit, 11 ft. 4 in. by 13 ft., and 40 ft. deep, to a drift 74 ft. from the circular shaft, to which it is connected. It will be recalled that this air shaft was sunk prior to the merger and before such a large development was contemplated. As a consequence, while the shaft is of ample dimensions for the requirements of the next 6 or 8 years, nevertheless, at some future date when the quantity of air required is considerably greater than it is today, a much larger shaft will be needed. When the larger air shaft is completed, a rock drift will be driven from the staple pit and the ventilation diverted. The fan is amply big for future demands.

As the intake air shaft is equipped with hoisting cages and is used during coal-hoisting periods by officials and also for the lowering of certain materials, it was necessary to build an air lock at the shaft mouth. This consists of two chambers, each 17 ft. 4 in. square. The section immediately over the shaft extends to a height of 33 ft. 10 in. above the surface. The roof is pierced with two small openings for the passage of the hoist ropes. The height of the outer chamber is 11 ft. 4 in. Between the two chambers there is a ¾-in. steel raising door, 10 ft. by 8 ft. 6 in., with a trap door 6 ft. 6 in. by 2 ft. The outer door is a double one, 8 ft. 6 in. by 6 ft. with a trap door insert. The air enters the shaft at a point 43 feet below the surface. At the shaft bottom, the air is split into two main airways. These run parallel to the main haulage for a distance of 2200 ft. At this point a connection is made with the main haulage road. One airway is continued to the face, the main haulage serving as a second main intake also. At the present time there is under construction an additional 13 by 11 ft. airway in an overlying seam. When this is completed the combined areas of the intakes will be 300 square feet.

The trolley haulage is operated on intake air only.

To prevent short circuit of the air to the coal shaft, an air lock was established on the main road, the doors being set 1250 ft. apart. This distance enables the motor trip to pass through the air lock without any appreciable reduction in speed. The doors are of wood, 7 ft. 4 in. by 6 ft. 5¾ in., bedded in concrete. A smaller door, 6 ft. 5¾ in. by 2 ft. 6 in., is set opposite the traveling way.

As the total pressure on the main door is nearly 700 lb., it is customary for the trapper (when the locomotive is approaching) to open the small door and equalize the pressure on both sides of the main door before attempting to open the latter.

The leakage is sufficiently great to ventilate thoroughly the haulage road from the air lock to the shaft. The air is split into five separate ventilating currents, so that the return air from each working section passes directly into the main return airway. The main return air divides naturally between No. 1 shaft and No. 1B coal shaft. At some future date the return to No. 1 shaft will be closed off, in which case all the return air will make its exit through No. 1B coal shaft via the dumb drift. The area of the airways leading to the shaft and the shaft itself is sufficiently great to do this without adding undue resistance to the air.

Fig. 9.—Door for control of ventilation in passage between Nos. 1 and 2 collieries.

It will be recalled that one object of the circular shaft was to provide an escapeway for the workmen in No. 2 colliery, in case of a disaster cutting off No. 2 mine shafts. Shortly after the circular shaft was sunk, a passage was driven between Nos. 1 and 2 collieries, 140 ft. in length and 6 by 8 ft. in section. In this passageway, doors were erected which effectively separated the ventilating currents of the two mines. These doors are referred to as "explosion doors." (Fig. 9.) It is hoped, in the event of an explosion in either mine, that the doors may withstand the force of the blast and provide an escapeway for survivors.

There are four doors, in pairs, two doors being hung on one frame. The distance between frame centers is 15 ft. 9 in. The frames are heavily constructed of cast steel and each is securely fastened into a heavy reenforced concrete ring. They are 23¼ in. thick on the bottom

and 18 in. thick at the top, leaving room between the doors for a man when the doors are shut. The battered frame insures the doors being shut by their own weight. The door opening is 2 ft. 6 in. by 5 ft. 6 in. The doors are also of cast steel 1¼ in. thick, heavily ribbed and further strengthened by having their outside surface convex, designed to withstand a pressure of 200 lb. per sq. in. The bearing surfaces between doors and frames are machined so that there is no air leakage between the mines. The door hinges are equipped with grease cups and open easily. The perimeter of the passageway between the doors and for a distance of 5 ft. on the outer ends is lined with concrete. A 4-in. pipe with a trap drains the water from No. 1B into No. 2 colliery.

This colliery is designed to hoist 2500 tons per 8-hr. shift. It was officially opened June 20, 1924. The maximum output raised in any one shift was 3204 tons, on Sept. 16, 1927.

Methods of Mining

Until 1923, the method of mining in the Sydney field was pillar and room, with pillar extraction under land areas and also in submarine areas under a minimum cover of 700 ft. Until recently rooms were driven 20 ft. wide and on centers varying from 48 to 64 ft., depending on the depth of cover. Crosscuts were driven 12 ft. wide on 87-ft. centers.

The coal is mined with Ingersoll-Rand radial cutters. These operate with a combined rotary and percussive action. Generally the inclination of the seams debars the use of shortwall machines.

It was customary for the machine runner and loaders to work on the same shift and a minimum of four rooms was allocated to each machine. Two rooms were cut on one shift, or approximately 40 tons. Two rooms were loaded out on one shift, making 10 tons per loader per shift. Crosscuts were mined and loaded out on the same tonnage rate as rooms and were only driven because they were necessary for effective ventilation.

Some time ago the width of rooms was reduced to 16 ft. and the width of crosscuts increased to 16 ft. The rooms were laid off so that the pillar length measured along the room plus 16 ft. was equal to the length of the crosscut. In this way each room had always two producing faces. As soon as a crosscut was driven through, the room had developed far enough to win another crosscut.

By setting the machine post in the center of the room or crosscut and cutting a full arc, the same tonnage per cut could be got as formerly with the 20 ft. width and the standard depth of cut of 5 ft. The machine runner was then given two rooms with crosscuts, and crosscuts being now equally as important as rooms, as producing places, this method doubled the concentration. In addition to this the following benefits were apparent:

1. Assured ventilation, since crosscuts had to be driven as soon as won.
2. Better roof conditions in rooms where the width was reduced by 4 feet.
3. Easier cutting for the machine runner, as one set-up of the machine was sufficient to complete a cut 16 ft. wide, while two set-ups were necessary to put in a cut 20 ft. wide.
4. Shorter flitting distances for the machine.

FIG. 10.—PRINCESS COLLIERY.

5. Easier hand loading, the throw being reduced by 2 feet. These benefits, although objected to by the men at first, are now accepted and the system is applied wherever possible. It is difficult now to get miners to leave a 16-ft. section and go to work in a 20-ft. room section.

In submarine mines the minimum cover under which any seam may be worked is 180 ft. This is governed by the Coal Mines Regulation Act. The company engineers have placed 200 ft. as the minimum cover

where the seam crops under the sea. Between the 180 and 700-ft. cover lines, supporting pillars must be left in. The extraction between these boundary lines averages 44 per cent. only.

Coincident with an increase in cover the roof shale became weak and in some cases the floor, which varies from fireclay to shale, also weakened and serious difficulty was experienced in keeping the chambers open. Even in narrow work development places it was necessary to support the roof with steel rails weighing 85 lb. per yd., used as booms and set at 1-ft. centers. Sometimes relief was got by driving an additional narrow place parallel to and slightly in advance of the main road. This place was allowed to collapse every 80 ft. or thereabouts and appeared to relieve the pressure somewhat on adjacent roads.

In Princess colliery (Fig. 10) where the seam is 5 ft. 8 in. high at a cover of 1300 ft. rooms were laid out to be driven a maximum distance of 200 ft. with the object of removing the pillars before excessive weight became evident. This did not prove successful; frequently the roof would collapse, shearing at the ribs, or the floor would give way and heave, almost entirely closing the place overnight. This necessitated ribbing in at considerable cost and even then a large percentage of the coal was lost.

Under these conditions it was necessary to change the system of mining and a start was made on the longwall method in 1923. A small section, 10½ South level, was started up, the cover at this point being 1430 ft. and the gradient of the seam 6 per cent. A breasting 100 ft. wide was carried forward, the face being on the dip and rise. Two roads were constructed in the gob, one being the haulage level and the other the intake airway.

A place developed at right angles to the level was brushed and packed as a main gate. The producing face was started up from the side of this gate forming an angle of 60° with the level. As the face advanced and the gate road was developed the face extended until it reached a length of 1600 ft. The face line coincided with the cleavage plane of the coal. Crossgates were turned off the main gate at 50-ft. centers. These were constructed parallel to the haulage level.

The brushing of the crossgates supplied ample material for pack building and being so close together the waste was practically all stowed. The coal was mined by handpicks and loaded into cars hauled to the face by horses. The method proved successful—the size of the coal was improved, roof troubles were eliminated and all the coal was extracted, thereby reducing the advance seawards proportionately.

The daily output from this section was 0.5 ton per ft. face. Later machine cut faces were laid out, these being at right angles to the level, each face was 250 ft. in length and two faces made tributary to one main level. The coal at the face was loaded on a jigger conveyor and delivered

into cars on the main level from the first face and to cars on a lateral gob road from the second face.

The elimination of the many crossgates, while effecting a saving in brushing costs, necessitated a radical change in the care of the coal face. The waste which had been almost solidly packed was now allowed to cave in. However, it was necessary to erect intermediate packwalls in the waste to prevent a sudden and extended rupture of the roof. These packwalls were built from waste tailings and were 12 ft. wide and set at 66-ft. intervals.

Three lines of hardwood chocks made up of 6 by 6 by 30 in. squared hardwood were erected parallel to the face, each chock being staggered with respect to the neighboring line. The chocks were set on 8-ft. centers on the dip and rise, the centers between lines being 7 feet. As soon as the third line was erected the rear line was withdrawn and the roof allowed to settle behind. The installation of conveyors and the limited brushing in this layout reduced the costs very considerably.

In Lingan district, longwall was started in 1924, in No. 16 colliery. The seam is 5 ft. 2 in. high and has an average pitch of 20 per cent. Overlying the seam is a weak shale varying from 2 to 14 ft. Above this there is a sandstone band varying from 20 to 45 ft. in thickness.

Operations commenced in No. 7 West level (Fig. 11), the cover being 1050 ft. As the levels had already been developed a considerable distance, the first operation was the removal of the level pillar and the widening out of the haulage level so as to erect stone-filled softwood packs, leaving a finished roadway 12 ft. wide.

A gate road was constructed at right angles to the level by widening out a headway which had already been developed.

A brushing of 6 ft. in height was carried in the level and fully 4 ft. in the gate.

The first face was 180 ft. in length and parallel to the gate. The coal was cut by a chain machine and loaded by hand on a jigger conveyor which delivered the coal into cars on the level. When the first face had advanced 80 ft., the second face was started, similar in all respects except that the face conveyor delivered the coal into a conveyor laid in a lateral constructed in the gob parallel to the level. This lateral conveyor delivered the coal into the main gate conveyor, which in turn delivered the coal into cars on the level. This made two loading points on the level which caused inconvenience and delays until the distance between these loading points became extended.

By the time the first face had advanced 170 ft. and the second face 100 ft. considerable difficulty had been experienced in keeping the faces open. Lack of experienced longwall miners and officials led to difficulties which otherwise could have been avoided. Few appreciated the importance of removing all rigid supports from the waste and substituting

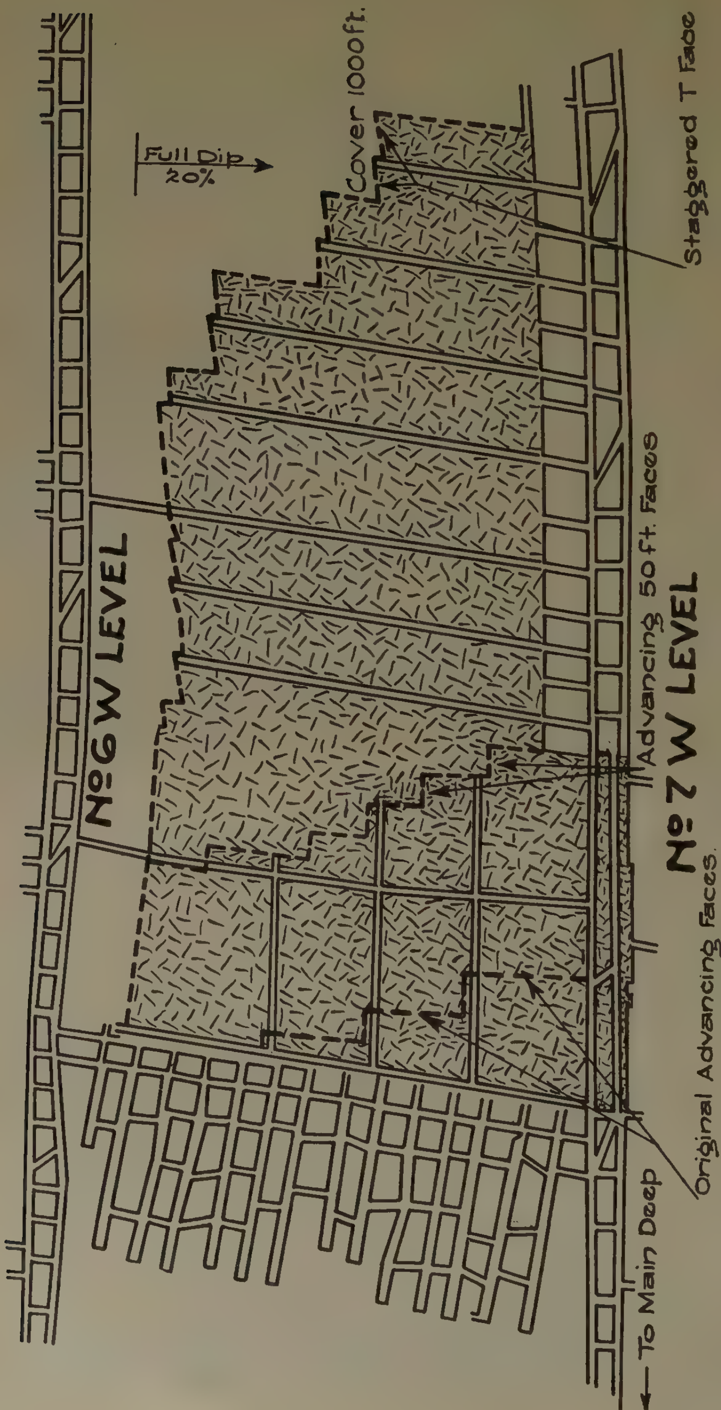


FIG. 11.—OPERATIONS AT No. 7 WEST LEVEL, No. 16 COLLIERY.

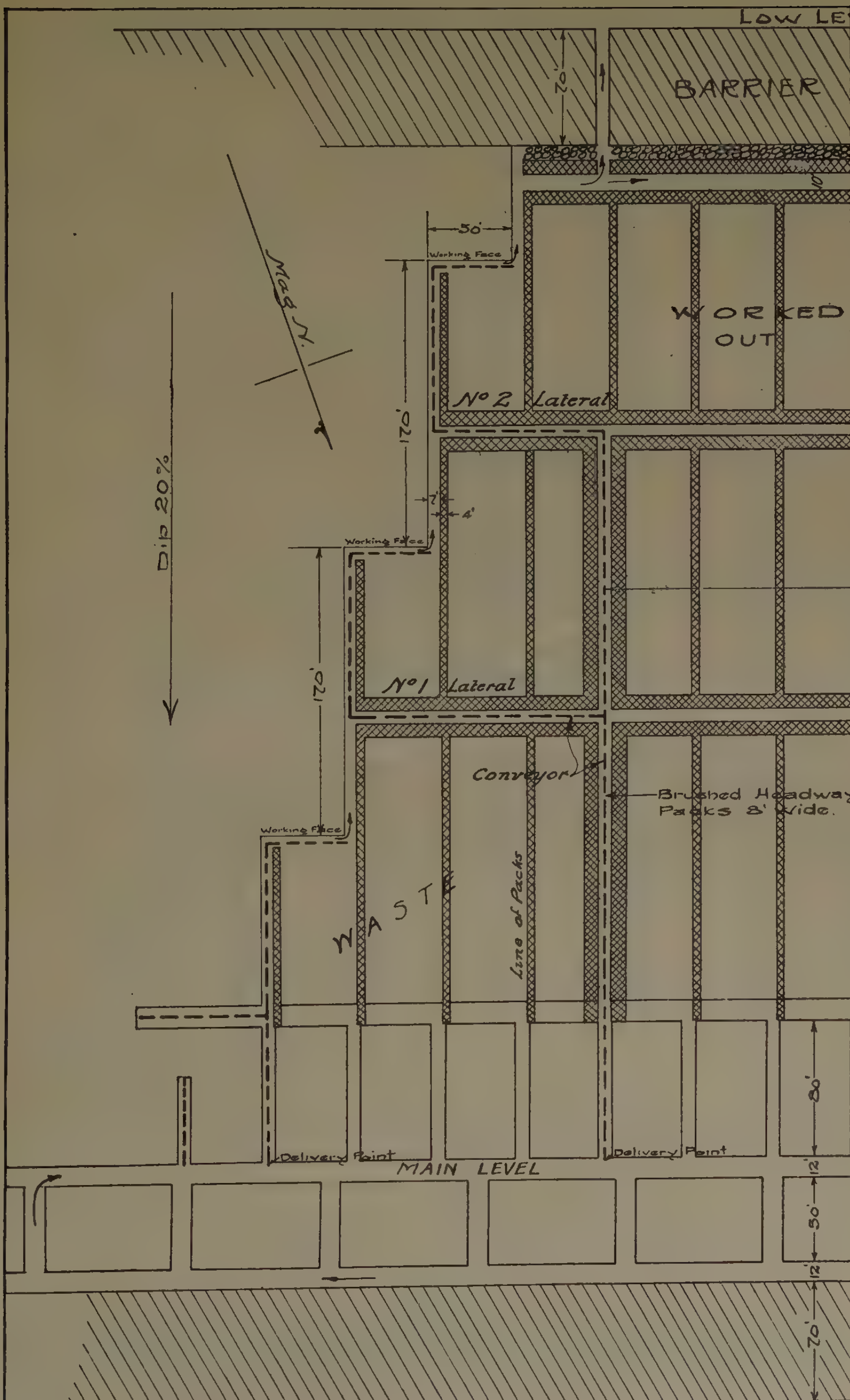


FIG. 12.—TYPICAL LONGWALL PROJECTION NOS. 14 AND 16 COLLIERIES; 50-FT. FACE ADVANCING LONGWALL.

stone packwalls to permit easy settlement of the overlying strata. The roof shale broke easily but the overburden from the sandstone upwards did not break readily. When it did break it frequently carried forward and closed the face.

After one of these closures it was found, when regaining the face by taking off a slab 12 ft. in width that the coal, if sheared in the tight corner, would fall readily from the face, provided the face line coincided with the cleavage plane, which was nearly on the strike. Advantage was taken of this and the direction of the faces changed to conform with it. Fig. 12 shows this clearly. A face 50 ft. in length was established and carried forward by shearing the tight corner. The coal was loaded on a face conveyor and delivered by it to a dip conveyor laid close to the solid coal and protected on the waste side by packs.

The coal from the first or lowest face was delivered by the dip conveyor directly into cars on the level.

When the first face had reached a point 20 ft. above No. 1 lateral gob road it was stopped and a new winning made from the level. The stopped face was started up again and carried forward as before, the only difference being that the dip conveyor delivered the coal into a conveyor on No. 1 lateral, thence to the main gate conveyor. When this face had reached a point 20 ft. above No. 2 lateral the coal was transferred through No. 2 instead of No. 1.

There were four 50-ft. faces in this section, the distance between laterals being 180 feet.

The faces were double-shifted and sometimes made an advance of 7 or 8 ft. per shift. This rapid advance led to difficulties. Hardwood chocks were set parallel to the face, the chock lines being on 8-ft. centers. Production in one day necessitated the erection of two lines of chocks. These were set by the miners. The withdrawal of chocks was done on the backshift by company men. If anything happened to prevent the complete withdrawal of the two rear rows of chocks, the close of the following producing day left the rear line of chocks 40 ft. back from the face. The removal of the rear row of chocks under these conditions was next to impossible, the roof having broken and fallen all around them so that they were buried in the gob. Apart from the waste of material, the buried chocks offered too rigid resistance to the settlement of the overburden and thus threw the weight forward, weakening the immediate roof in the vicinity of the face and frequently resulting in closure.

There was considerable difficulty in getting material to the face, necessitating three transfers, which added considerably to the cost.

While the jig conveyors were capable of handling all the coal on the inclines, it was found that the lateral jig conveyors, laid practically level on a gobbed road subject to heave, which threw them out of vertical alignment, were not capable of taking care of the coal that could be

mined at the face. Where the laterals were short and before the floor had started to heave, a rapid advance of the face could bo depended on;

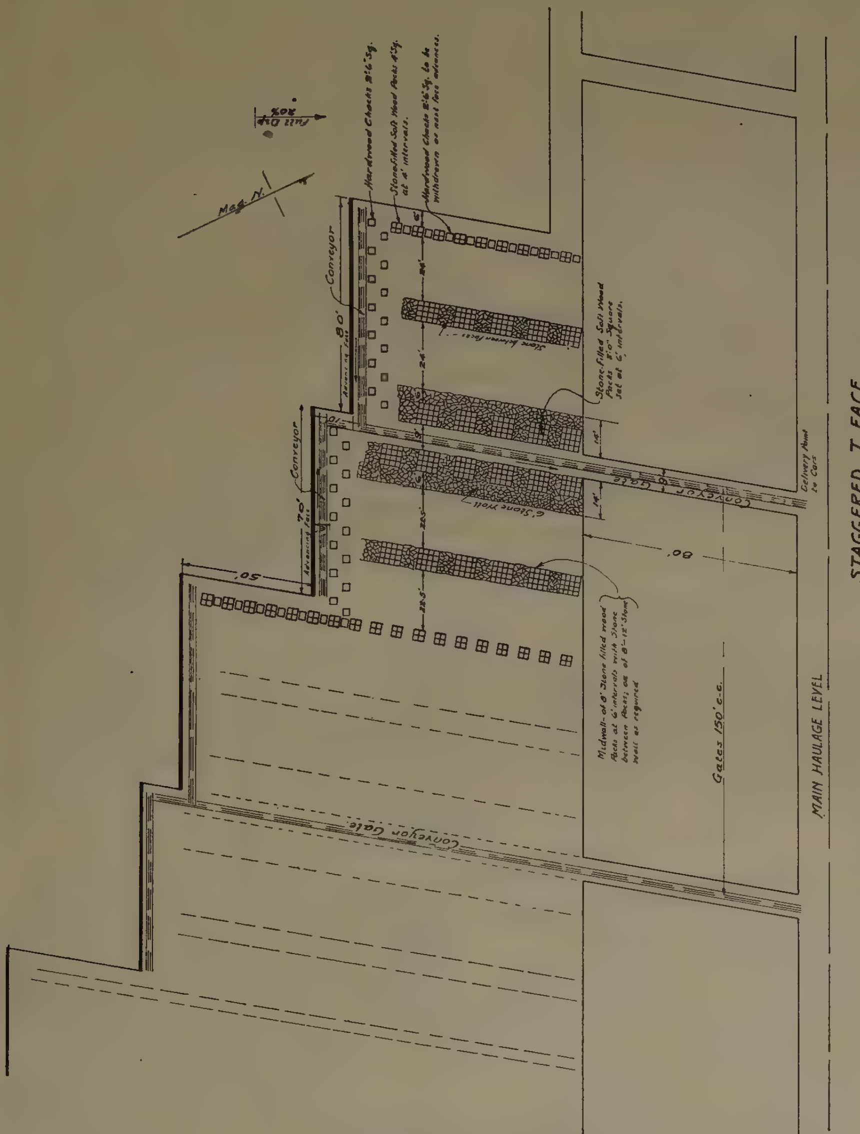


FIG. 13.—PROPOSED SYSTEM LONGWALL EXTRACTION, COLLIERY 16; STAGGERED T-FACE ADVANCING LONGWALL.

when they became extended about 80 ft. and were subject to heave there was a slowing up of the face, which was not satisfactory either to the contract miner or the colliery manager.

A later development is shown by Fig. 13. Here a gate is constructed on the rise of the seam with a producing face on either side, each approximately 75 ft. in length.

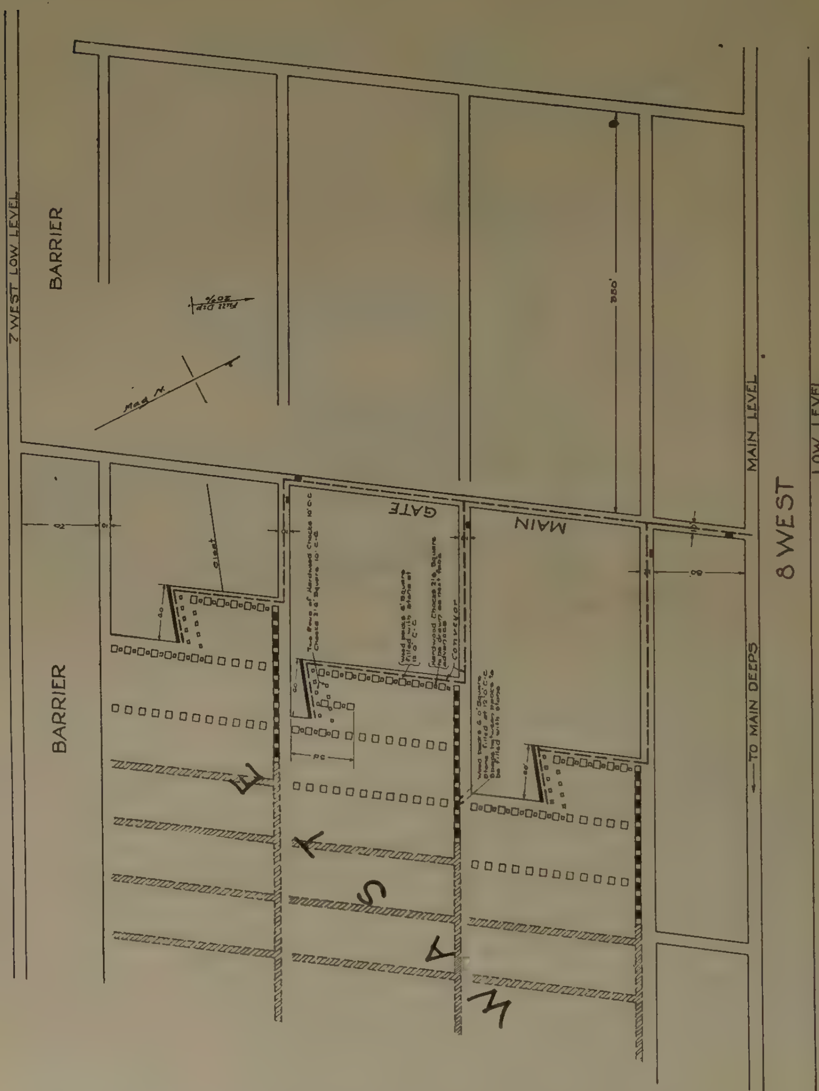


Fig. 14.—No. 16 Colliery; proposed method of longwall retreat, No. 8 W. section; 50-ft. face retreating longwall.

The east face is carried about 10 ft. in advance of the west side, a shearing being established in line with the west side of the gob road and another shearing in the west corner. Stepping the faces permits delivery from the face conveyors to the main conveyor without crowding the pan at one point.

Fig. 14 shows a retreating system developed to avoid brushing. The lateral conveyors are more stable, being laid in roadways in the solid; however, the cost of narrow work development offsets to some extent the savings effected by elimination of brushing. This system is applicable in limited areas, *i. e.*, where the floor is reasonably firm.

Total extraction in the submarine area of this field has been carried out successfully where the seam was 6 ft. 5 in. high and the overburden 700 ft. In all probability total extraction without stowing could be carried out safely with less cover but it is felt that it is better to err on the safe side, especially where an error in judgment would be so disastrous.

What is of more importance than depth of cover is the character of the overburden. Where yielding shales and clays predominate the depth of the overburden may be safely reduced; conversely, where limestone and sandstone predominate the cover must be increased for the same safety factor. Undoubtedly where a break does occur, the softer shales and clays in time puddle the fracture and render it impervious to the flow of water. In the drawing of pillars it has happened that an inflow of sea water was encountered. This lasted for several months but gradually diminished and then apparently ceased.

Subsidence of the overburden extends over a long period of time. Unfortunately records were not kept in the past to determine the amount and rate of subsidence, but observations in a few cases have shown subsidence to be equal to 60 per cent. of the extraction over a period of 15 years, at a depth of from 400 to 600 ft. Total extraction of two or more seams superimposed has not been attempted so far; however, the conclusion reached by T. Forster Brown, a British engineer of wide experience in submarine mining, is that total extraction in this field may be carried out safely allowing 100 ft. of strata for every foot of extraction.

Ventilation

Undoubtedly the ventilation of a submarine mine is a problem of prime importance. Granted suitable grades and a moderate overburden, coal may be mined and transported economically an indefinite distance underground. Ventilation of remote workings, however, is a problem that demands the closest attention and study of the mining engineer.

From the time mining commenced in this field down to the close of the 19th century little thought was given to the question of submarine mining, except for limited distances. This is reflected in the attitude of the operators in the Sydney Mines district, who as late as the year 1900 did not deem it worth while to secure a submarine lease farther than 1 mile from the shore. Evidently this was considered the limit of profitable mining, although at the present time the workings there have actually reached a distance of 2½ miles from the shaft. With this attitude of

mind it is not to be wondered at that provision for the ventilation of workings 2 to 3 miles from the shore was not made.

Pillars were drawn in the area worked over which extended fully 1 mile from the shore in 1913, and although at that date it seemed feasible to mine a much greater distance seawards, nevertheless it was recognized that ventilation would be the limiting factor. It was impossible to construct airways through the waste. Airways might have been constructed in a stratum under the seam but the cost would have been prohibitive. In 1922 the situation became acute and it was decided to install a booster fan on the main return at a coursing distance of 21,000 ft. from the upcast shaft.

The surface fan is a motor-driven Capell, 20 by 5 ft., which exhausts 51,000 cu. ft. of air per min. at 2 in. water gage. The booster fan, which is driven by a 12 by 15 in. Robb engine, using 700 cu. ft. of compressed air, is of the Sirocco type. This fan exhausts 45,000 cu. ft. of air with a 2-in. water gage. The booster fan operates continuously except on week-ends and idle days.

Although serious objections may be raised against the normal ventilation of a mine being made dependent on an underground booster fan, nevertheless, under the circumstances, this appeared to be the most feasible solution. The ventilation of the faces is satisfactory, the percentage of methane in the returns being negligible.

Fig. 15 shows the position of the intake and return shafts; also the location of the main and booster fans and the direction of the air currents. At the point *A* there is a pressure tending to cause leakage from the intake to the waste, which is the return. At *B* there is a pressure in the return tending to cause leakage from the return into the intake. The booster fan near *B* creates a depression of 2 in. w.g., measured at the fan, which induces a flow into the mine at the intake shaft. At point *A* which is close to the intake shaft the absolute pressure on the air is practically atmospheric, $14.7 \times 144 = 2116.8$ lb. per sq. ft. On the intake at *B* this is considerably reduced by the pressure absorbed between *A-B*, due to resistance to the flow of air in the intake between these points.

The air is split below *B* and ventilates the working faces, joining together again at the booster fan. Since the depression created by this fan is 2 in. or 10.4 lb. per sq. ft., the absolute pressure of the air on the intake side of the fan is $2116.8 - 10.4 = 2106.4$ lb. As the fan develops 2 in. of w.g. the air on the delivery side of the fan is restored to atmospheric pressure. This being in excess of the pressure on the air in the intake at *B* tends to cause a flow from the return.

The main fan at the return shaft draws the air from the booster fan and also creates a depression at the surface of 2 in.

There is a steady fall in the absolute pressure of the return air from *B* to the main fan due to resistance. It is obvious that the absolute pressure

of the return air is less at A than the absolute pressure of the intake. Leakage into the return necessarily follows, unless thoroughly sealed.

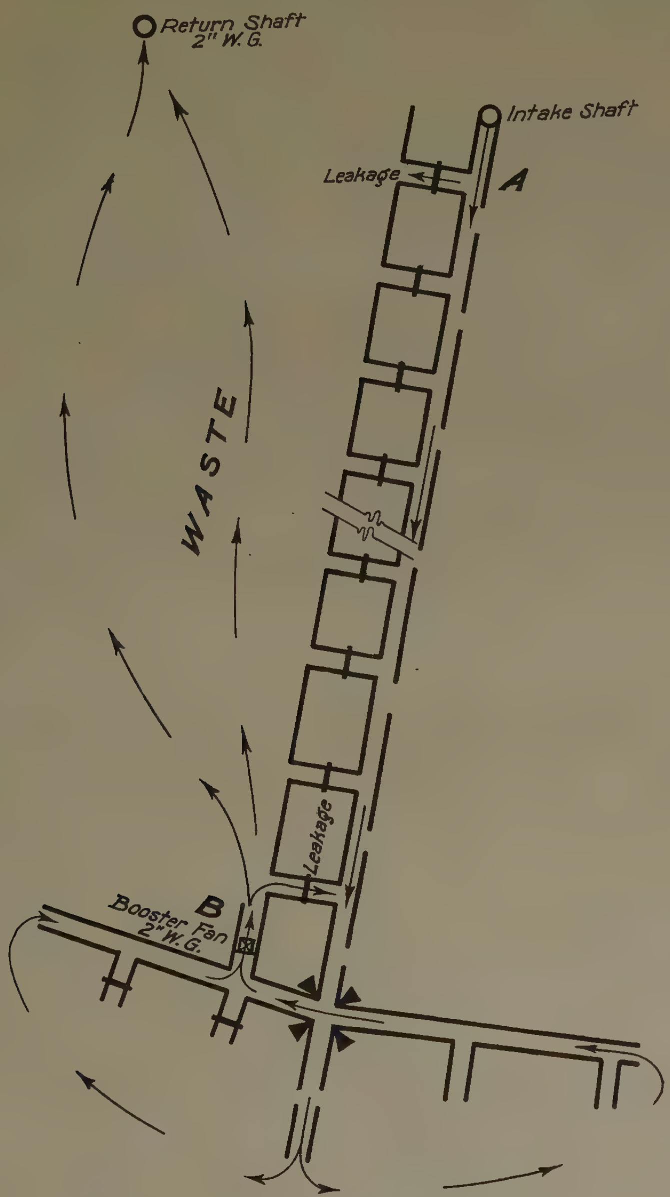


FIG. 15.—INTAKE AND RETURN SHAFTS; SHOWING ALSO LOCATION OF MAIN AND BOOSTER FAN AND DIRECTION OF AIR CURRENTS.

For effective and economical ventilation it is always desirable to build stoppings as tight as possible. It will be seen, however, that

leaky stoppings are a much more serious matter when booster fan ventilation is in force, inasmuch as there is a possibility of fouling the intake by recirculation.

No consideration has been given here to the difference in elevation of points A and B, or the temperature of the air in the workings. This, however, merely complicates the problem without making any clearer the point emphasized, namely, the necessity for tight stoppings.

Prior to the war it was realized by the Dominion Coal Co. engineers that steps would have to be taken to assure adequate ventilation for future submarine mining. No action was taken, however, until the strenuous war years had passed. In 1921 the problem was studied and it was decided that steps should be taken immediately to establish satisfactory ventilation that would be adequate to meet requirements for at least the next 20 years. The following principles were laid down: (1) ventilation to be ascensional; (2) two or more main intakes for each mine; (3) air to be split as near the shaft bottom as possible and secondary splits laid out to maintain normal velocities around the working faces; (4) definite returns to be constructed contiguous to the intakes or through the waste; (5) air charged with an excess of 0.5 per cent. methane not to be allowed to enter a working section but to be conducted directly into the main return; (6) haulage and traveling roads to be made neutral as to air; (7) construction of main airways to be of a permanent character. This entailed the construction or rehabilitation of 80 miles of main airways, the erection of 85 overcasts and 2100 concrete stoppings, at a cost in excess of $3,000,000. The work of rehabilitation was started at once and pushed ahead vigorously.

Ventilation—No. 2 Mine

A description of the ventilation and airway construction of No. 2 colliery may be taken as typical of the submarine mines generally. This mine is ventilated by a 20 ft. by 7 ft. 6 in. Walker fan, designed to deliver 400,000 cu. ft. of air per min., against a 4½-in. w.g. The fan is rope-driven from a 30 by 48 in. Corliss valve engine. At the present time it circulates 175,000 cu. ft. with a water gage of 5 inches.

There is also a steam-driven, direct-connected Dickson fan as a stand-by, capable of delivering 250,000 cu. ft. of air per min. These fans are forcing, as are the majority in use in this field. However, all are installed so that the air current may be readily reversed.

The air is delivered through a 12 by 12 ft. concrete air duct, below the surface, to an air shaft 10 by 17 ft. and 855 ft. deep. At a point 40 ft. from the bottom of the shaft the air is split; an inclined cross-measures drift, 12 by 12 ft., driven over the seam and tapping the seam 200 ft. from the shaft, forms the south side airway. At this latter point the air is divided, entering two parallel intake airways, each 12 by

6.5 ft. These are continuous to the face of the South Deeps—12,000 ft. distant. This split is again divided so as to ventilate the workings north and south of the South Deep separately. There are also separate returns to these splits, joining together near the coal shaft, which is the main return.

The quantity of air entering the south split is 68,000 cu. ft. The combined quantity entering the secondary splits is 53,000 cu. ft., showing a loss by leakage of 15,000 cu. ft. or 22 per cent. in a distance of 2 miles.

The main north split begins at the bottom of the air shaft and traverses a single airway for 300 ft., here the air enters two airways, one on the north and the other on the south side of the haulage deep. These have a combined sectional area of 150 sq. ft. At a point 1½ miles distant, the air on the south side is deflected across the haulage road by means of overcasts and from this latter point to the face of the deeps, a distance of 6000 ft., the airways are parallel and 50 ft. apart.

The main north split is divided into four secondary splits. The quantity of air entering the main split is 105,000 cu. ft. and the combined quantity entering the four splits is 78,000 cu. ft., the loss through leakage over a distance of 2½ miles being 27,000 cu. ft. or 26 per cent.

It must be noted that the leakage loss is not a function of the distance. Construction is being carried on at various points on the airway, necessitating the installation of many doors for the purpose of handling material and removing muck. When this work is completed and the doors bricked up the loss from leakage will be reduced. The air in the main returns carries 0.34 per cent. methane.

Owing to the weak roof it is necessary to give ample support. Figs. 16 and 17 show typical construction of the main intakes. Eighty-five pound re-layer steel rails are used as booms. Although not designed to serve as beams, nevertheless because of their permanent character and their greater strength compared to spruce wood booms, they are used in all main roads.

The varying temperature of the intake air readily disintegrates the soft roof shales and necessitates close lagging. Frequently the sides have also to be lagged as the coal is friable and spalls readily if the rib of the airway coincides with the plane of the coal.

Usually the return airways are timbered with props and caps. These are barked and given a hot salt bath for at least 24 hr. as a preservative. As the temperature of the return air is more nearly constant, the roof does not disintegrate so readily as in the intakes, but the return air hastens the decay of the wood. This is more evident in some other collieries than No. 2.

Fig. 16 shows also the method of construction of return airways recently put into effect where the return air is particularly destructive of wood.

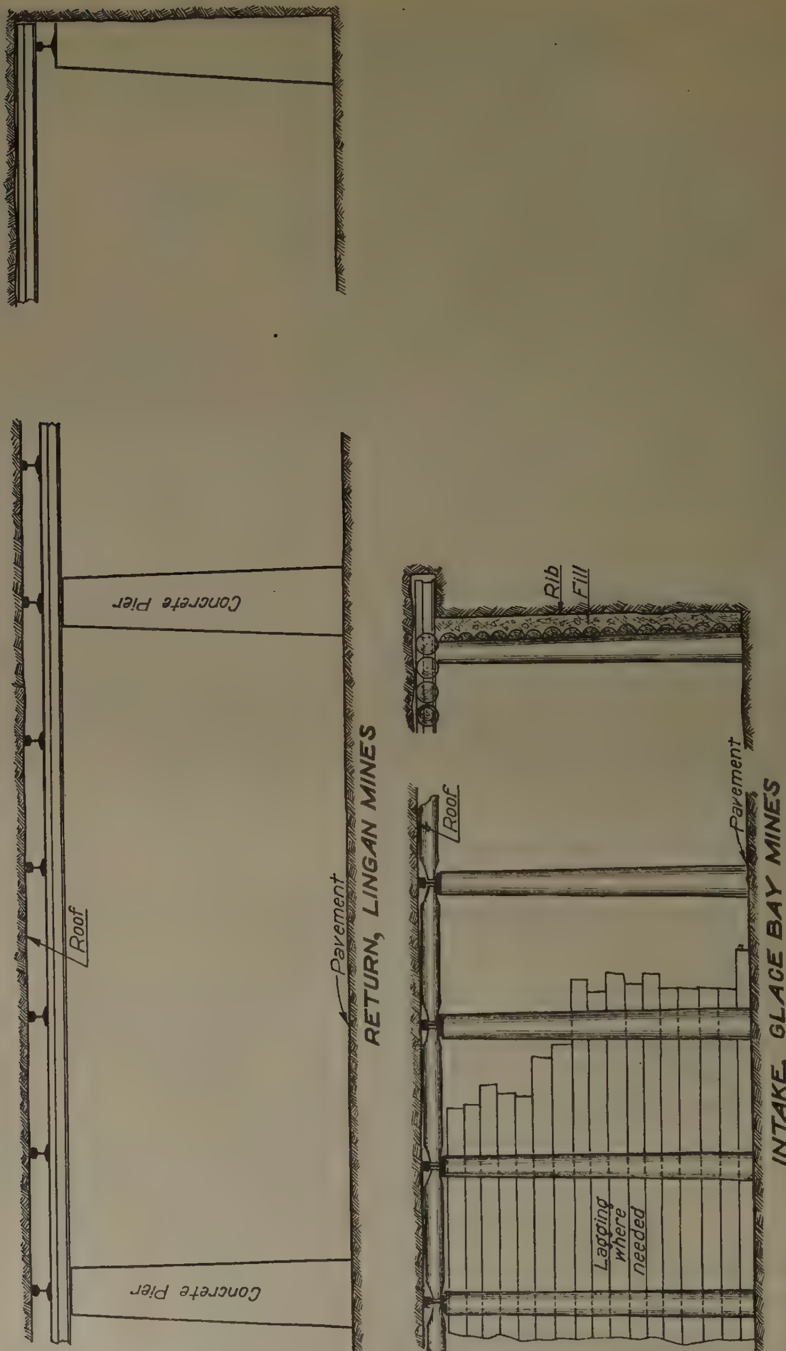


FIG. 16.—TYPICAL CONSTRUCTION OF MAIN INTAKES.

Fig. 18 shows a typical concrete stopping built only in main airways, and the type of stopping used in subsidiary airways, the plaster mixture used being 1 hard wall plaster, 2 cement, 4 sand.

A standard ventilation door is shown on Fig. 19.

Fig. 20 shows a typical permanent overcast. Temporary overcasts are much ligher in construction, although also of concrete.

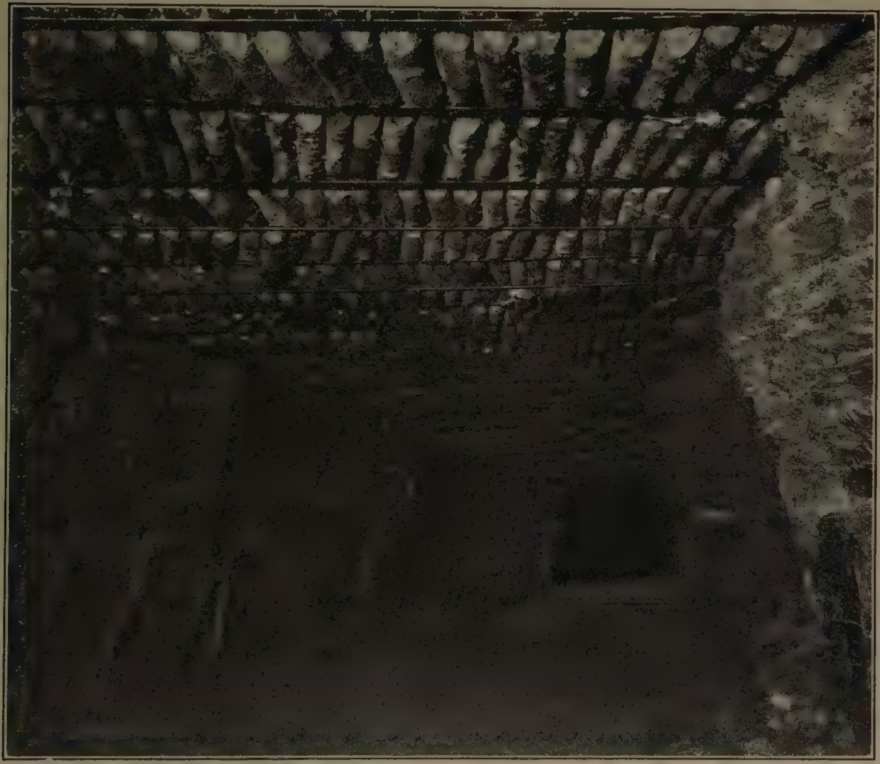


Fig. 17.—Typical construction of main intakes.

Regulators are avoided wherever possible; when used they are generally placed at the mouth of the split to reduce leakage losses.

A careful check is kept of the quantity and condition of the air in each split. There is a staff of ventilation engineers for the entire coal field. These men direct airway construction and make recommendations to the colliery manager on rearrangements of the air splits when desirable; they also measure and sample the air at least once a month, or oftener as the occasion demands. In this way a constant check is kept of the condition of the ventilating currents.

The problem of future ventilation will be solved in great measure by constructing airways of large sectional area. It is not suggested that the

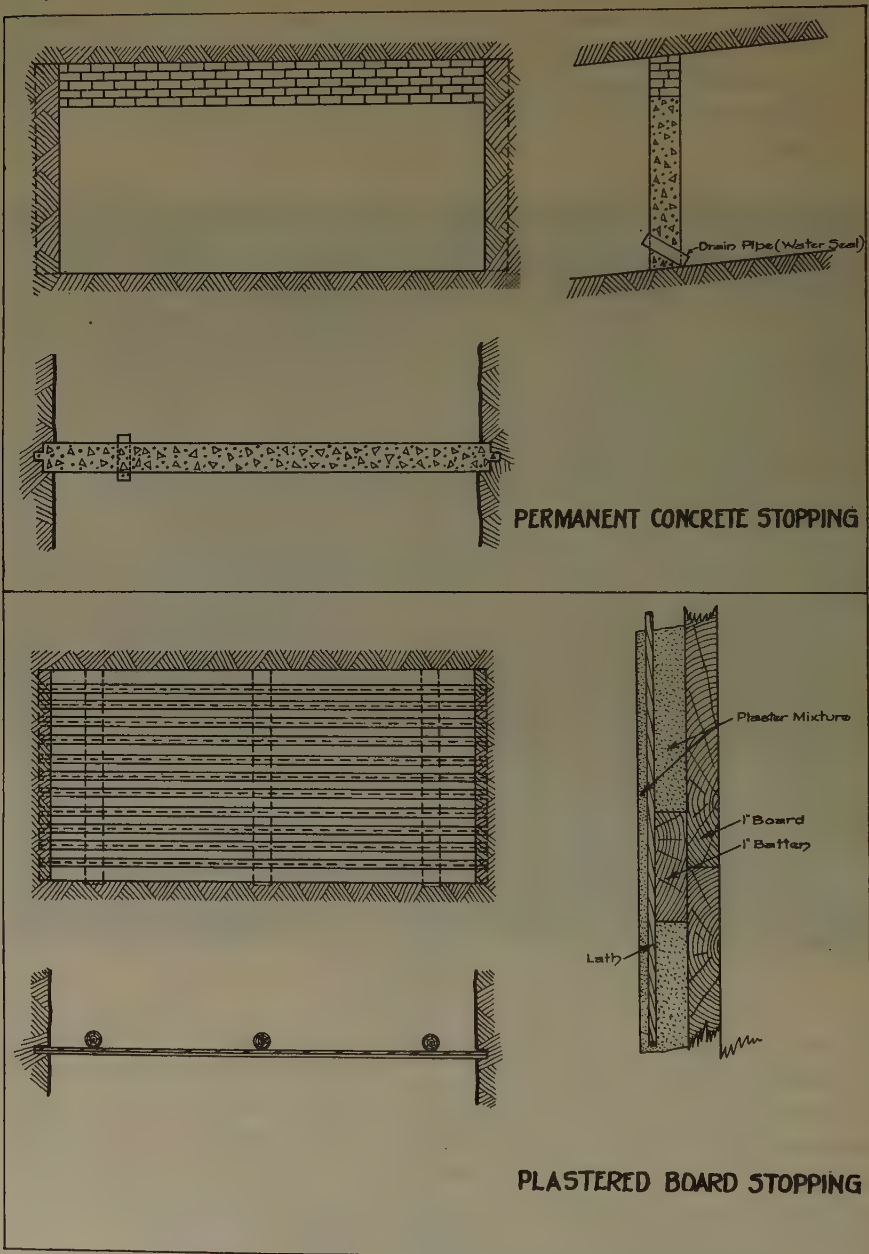


FIG. 18.—TYPICAL CONCRETE STOPPING BUILT ONLY IN MAIN AIRWAYS.

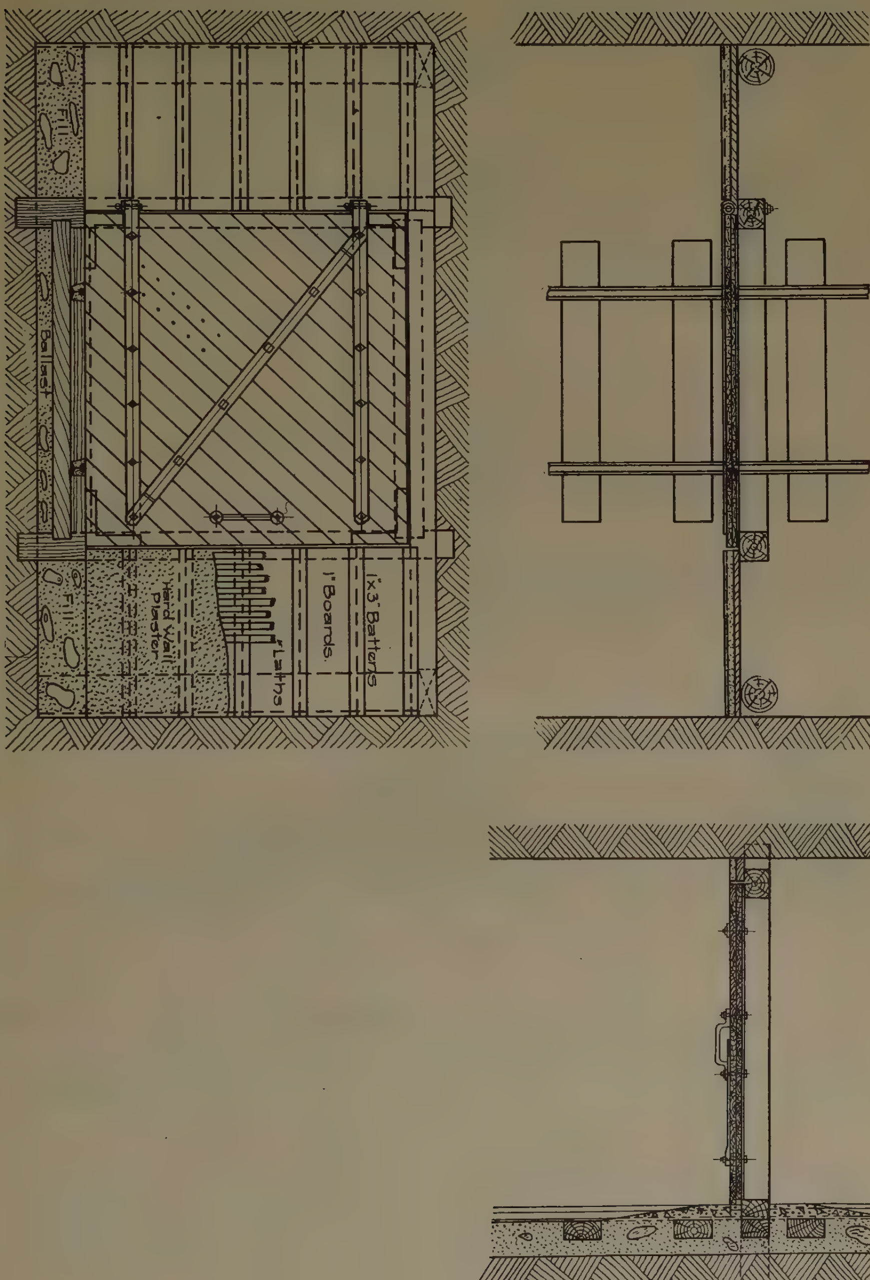


FIG. 19.—TYPICAL MINE DOOR. DOORS PLACED WHERE THE PRESSURE EXCEEDS 1 IN. WATER GAGE HAVE SIDES BUILT OF CONCRETE IN PLACE OF PLASTER.

airways being constructed at the present time will be adequate 30 or 40 years hence. We believe, however, that they will take care of the requirements for the next 20 years and that the present expenditure is justified on this basis.

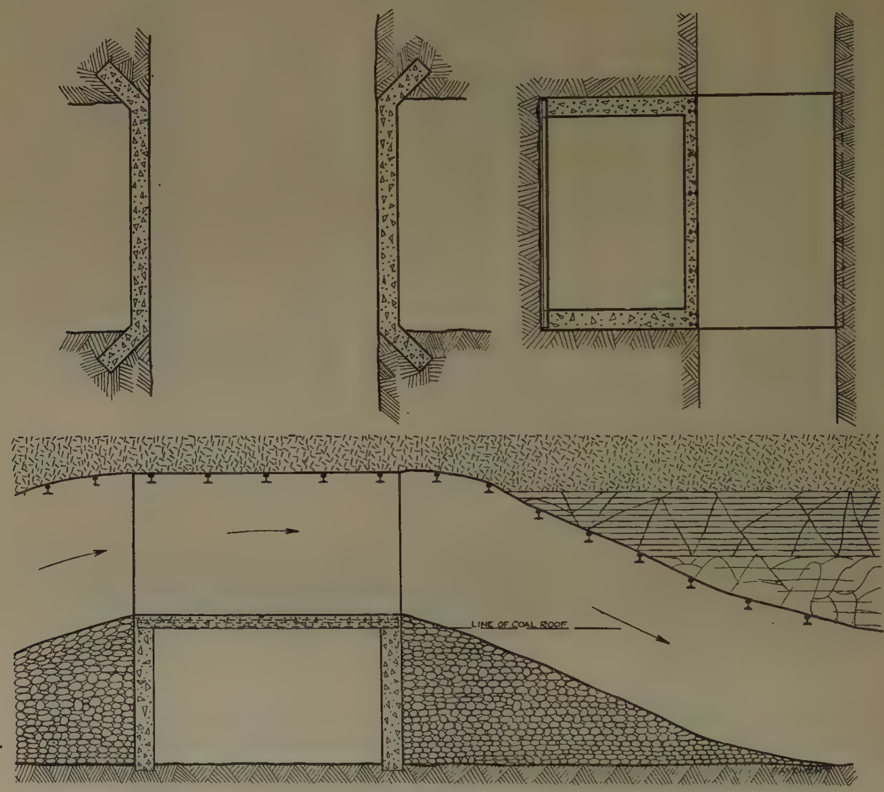


Fig. 20.—Typical overcast.

Underground Transportation

In a submarine mine where it is impossible to cut off an extended haulage by means of another shaft, the question of haulage is necessarily important.

All the seams in the Sydney coal field dip in a seaward direction, varying from 6 per cent. on the axis of the Glace Bay basin to 22 per cent. on the Lingan basin. These inclinations for extended distances prohibit the use of locomotive haulage. In general, the system in use where the inclination is less than 10 per cent. is endless rope and where the inclination exceeds this, single rope trip haulage, hoisting in balance.

If reference is made to Fig. 2, it will be noted that No. 2 colliery shaft is located on the hub of the Glace Bay basin, the contour of the seam

forming an arc north and south of the shaft; also that No. 1B colliery is located on the south side of an anticline while the greater part of the area tributary to this shaft is north of the anticlinal fold. These two mines are exceptional in this respect and are suitable for some type of locomotive haulage.

All the coal in No. 2 to the rise of the shaft was lowered to levels following the contour, and compressed air locomotives hauled the output along these levels to the shaft. When the workings had developed to the dip of the shaft the main endless rope system was established and the locomotives discarded. The trolley locomotive haulage in No. 1B has been detailed in the description of that colliery.

The features favoring endless rope haulage are: (1) low speed of movement of cars; (2) steady output with short landings, the mine cars being delivered singly, and (3) less danger of derailment and less damage resulting when derailment occurs. The disadvantages are: (1) the necessity of maintaining a double-tracked road throughout the entire length of the haulageway; (2) increased labor in handling cars at the landings; (3) cost of maintaining pulleys and ropes and (4) greater number of cars required for a given output.

With an extended haul and the number of landings kept to a minimum there is no doubt that a steadier and greater output is assured than with single rope plane haulage and this more than offsets its disadvantages. The longest haulage of this type in use in this field is in Caledonia mine, the length of haul being 16,000 ft. over a gradient of 7.33 per cent. against the load and the output 1300 tons per 8-hr. shift.

Up until 1922, this system was operated by single reduction spur gearing, driven by a 22 by 42 in., noncondensing slide valve steam engine, built in 1872. This engine is still in use and giving excellent service, the only major repairs necessary in its life of over 50 years being the replacement of the piston rod and the reboring of the cylinder. At that date, however, additional load made it necessary to increase the engine power and strengthen the gears.

Rather than scrap the existing engine, it was decided to install an auxiliary. This consisted of a fourteen 24 by 42-in., compound, noncondensing, slide valve Matheson engine, coupled to the extended end of the existing crank shaft, the two engines having a common fly wheel.

The present gears consist of a cast steel driving drum, 8 ft. 3 in. dia., lagged with cast iron segments, bolted to its periphery. On either side of the rope wheel is cast a brake path 7 ft. 2 in. dia. and 6¼ in. wide. The gear wheel is of cast steel with 96 cast teeth, 3¼ in. c.p. and 15 in. face. The gear and the rope wheel are keyed to a 14-in. dia. shaft which is nicked down to 10 in. dia. in the bearings. The pinion is also of cast steel with 16 fully shrouded cast teeth. The friction clutch is of the

band type, engaging on a wheel 5 ft. 6 in. dia. keyed to the extended hub of the pinion.

The mine car has a capacity of 1.4 long tons and is equipped with 12-in. cast steel wheels and plain bearings. The over-all dimensions are length 8 ft. 1½ in.; width, 3 ft. 7⅜ in.; height, 3 ft. 6⅛ in.; weight 1450 lb.; gage, 23½ inches.

The haulage rope is 1¼ in. dia., best plow steel, Lang's lay, with 6 strands of 7 wires over a hemp core, weighing 2.45 lb. per ft. The breaking stress of this rope is 134,000 lb. The tension developed when the rope is fully loaded is 75,000 lb. It will be seen that the safety factor of this rope is very low. To increase the size of the rope would lead to pulleys of very large diameter unsuited to underground conditions, and further an abnormal share of the power would be required for movement of the rope itself, which even at the present time accounts for almost 20 per cent. of the rope stress.

It is considered that in this instance the economical limit of a single endless haulage has been passed and it is proposed to install next year a motor-driven relay endless system at a point 2½ miles from the shaft.

In the Waterford collieries where the inclination averages 23 per cent. the trip haulage system obtains. The main haulage engine at No. 16 colliery, which is a slope mine, consists of two drums, 8 ft. dia., 4-ft. face, capable of holding 8000 ft. of 1⅛ in. rope, operated through friction clutches from a geared slip ring motor 1200 hp., 6600 volt, 375 to 370 r.p.m., controlled through a liquid rheostat.

The over-all dimensions of the mine car are: length, 6 ft. 11 in.; width, 3 ft. 7 in.; height, 3 ft. 3 in.; gage, 2 ft. 6 in.; weight, 820 lb.; average capacity, 0.89 ton. Trips are hoisted in balance, 25 cars per trip, the rope speed being 18 miles per hr. The rope stress is 18,000 lb., giving a safety factor of 6. The economical limit of this type of haulage in this district is about 1¼ miles.

A haulage of the same capacity, and quite similar, is about to be installed underground in No. 1B colliery at a point 10,000 ft. from the shore. This haulage will necessitate an excavation of 36 by 40 ft. by a height of 20 ft. and is the largest underground haulage installation known to the writer. Under the conditions there it is anticipated that this installation will be capable of hauling 2500 tons from a distance of 8000 ft. in 8 hours.

Subsidiary haulages are of the main and tail or single rope types operated by compressed air. These, however, are gradually being replaced by electrically-driven haulages. Horses are used only in gathering from the faces to room landings.

Surface Transportation

The Dominion Coal Co., Ltd., owns and operates the Sydney & Louisburg Railway, for the purpose of collecting the output from its

collieries and assembling it at its shipping piers. The main and branch lines have a combined length of 79 miles and the yard tracks aggregate 50 miles. The rolling stock consists of 30 locomotives, ranging in weight, on drivers, from 75,000 to 174,000 lb. There are 1800 coal cars. The monthly tonnage hauled is about 400,000 or 5,400,000 coal ton miles.

The principal shipping piers are at Sydney and Louisburg; the former for summer shipping, the latter, being an open port throughout the year, is used in winter. Four steamers may be loaded at Sydney, and two at Louisburg, simultaneously, at the rate of 1000 tons per ship per hr., the record being 7000 tons loaded in 5 hr. in one ship.

Power

Most of the collieries were originally laid out to be operated by individual steam power plants, due in part to the established practice of separate companies from which the Dominion Coal Co. was formed. Consequently the history of the power system is that of a slowly centralizing process and of a gradual and continuing change from steam to electricity.

The increase in use of electric current has almost doubled during the last 5 years, the present annual consumption of electric energy being in the neighborhood of 55,000,000 kw-hr., although we still have in use steam boilers at the collieries having a combined rated capacity of 10,000 boiler horsepower.

There are two main generating stations, one in Glace Bay and the other at Waterford, the former being the first installed and now becoming more or less obsolete, the ratio of production of energy being 19 and 81, respectively.

The Waterford power plant consists of three Babcock & Wilcox boilers and 4 Bettington pulverized fuel boilers, having a combined capacity of 6800 boiler hp., supplying steam at 175 lb. pressure to an 8100-kva., 6600-volt, 0.9 power factor turbogenerator as the main machine and a 6000 kva. stand-by generator, generating three-phase current at 25 cycles.

To replace the Glace Bay station and to provide for further electrification of the mines, the company is now contemplating the erection of a power plant of 20,000 kw. capacity.

Mechanization

Owing to the inclination of the seams, a weak top and the gaseous and dusty nature of the coal, mechanization at the face offers serious difficulties. All the operations in the mines are performed mechanically, except loading, this of course being the last step in mechanization.

Many types of mechanical loaders have been investigated but a design to meet the needs of this field has not yet been found. At the

present time experiments are in progress with a type of conveyor and loader which seems suited to the conditions.

The exclusive use of compressed air at the working face, entailing as it does large and cumbersome machines, compared with the smaller and more compact electric motors operates against the ready application of mechanical loaders and conveyors. Undoubtedly as operators and workmen become more familiar with the use of electricity underground its application will be extended to the working face.

The laying of coal dust incident to face-working by means of sprinkling and further precautions by means of stone-dusting would, in our opinion, render such installations safe. On the other hand, it must not be forgotten that the inclination of the seams and the necessity for close timbering does not give the electric machine the advantages it has in flat seams and where good roof conditions obtain.

Reference has been made in this paper to a line 3 miles seaward from the shore. This is not necessarily the economic mining limit, although the writer does regard this as approximately the economic limit of our present mines. To win an extended area beyond this, say from 3 to 6 miles from the shore, does not offer insuperable engineering difficulties, although at the present value of coal it is not economically feasible to do so.

The method of approach would probably be by deep land shafts and long cross-measures drifts intersecting the most valuable seam at a distance of 5 miles from the shore. Such a mine would of necessity be a large producer and would require a sea frontage of at least 3 miles on each side of the point of entry by the drifts.

If the shafts were sunk at a point where the Phalen seam has a cover of 500 ft. and the drifts given a rising grade of 4/10 per cent., the drifts would intersect the Phalen seam with about 2100 ft. of cover and necessitate the sinking of the shafts to a depth of 2200 feet. This operation would make possible the winning of the Phalen, Harbor and Hub seams, the two latter overlying the Phalen at 450 and 830 ft., respectively. The output would be 8000 tons per shift or 16,000 tons per day. Transportation would be by trolley locomotive haulage and would necessitate the employment of probably eight main line locomotives. The overburden would not be excessive, the method of working could only be determined when the seams were reached, but it would be some type of longwall.

The most difficult problem would be that of ventilation. For a colliery such as is outlined above, the quantity of air required would be 500,000 cu. ft. per min. If this were passed through two intakes and two returns each 250 sq. ft. in section, the resistance would equal 17 in. of water gage. With the air split at the entry to the seam the resistance of the inside workings would in all probability not exceed

6 in., giving a total resistance of 23 in. It is evident that in addition to an exhaust and forcing fan at the return and intake shafts, respectively, another booster fan at the entry to the seam would be necessary to keep down the pressure on the air to workable limits.

The use of liquid air both as a cooling agent and as an auxiliary to the regular ventilating current may offer possibilities when long distance submarine mining becomes necessary in this field. At the present time the cost would be prohibitive.

Very little study has been given to this subject. It is primarily an economic rather than an engineering problem.

DISCUSSION

R. D. Hall, New York, N. Y. (written discussion).—The difficulties in controlling a marly roof are well illustrated by the author. It is to be regretted that he did not find opportunity to dilate on the intricate mechanics of the roof action which such an expansive material causes. Perhaps I may be pardoned for discussing this matter at some length, for expansive materials are found in many roofs, and little or no study has been devoted to them. An inquiry into this quality of mine roofs and its effects might be conducted with advantage.

In some roofs, such as those at Gallup, N. M., and at Dawson in the same state, the expansive quality of the roof produces extraordinary phenomena which need not be discussed here. I cannot refrain, however, from making reference to the experiments in Great Britain which have shown that even coal expands sometimes as much as 2 per cent. when it is wetted.

If the lower part of the roof on being exposed to the mine atmosphere should expand over the pillars it would raise the roof against its own weight unless the pillars would crush or embed themselves in the floor or do both. Moreover, though the lifting action would be confined to the area mined it is possible to conceive that the roof would be raised over both the unmined and mined areas (Fig. 21). This however is unlikely. It is more likely that the roof would be flexed upward or "hogged" (Fig. 22). The raising of the roof and the flexing arising from that action would bring a heavy load on the pillars. It must be remembered, however, that the vertical expansion is at a maximum at the pillar edges and at a minimum in the center of the pillar. The edges expand and the swelling being hindered by the pressure of the roof causes the rock to bulge or bend into the space above the roadway, to which movement the only resistance is the inherent strength of the expanding rock. The lifting of the roof is thus made problematical.

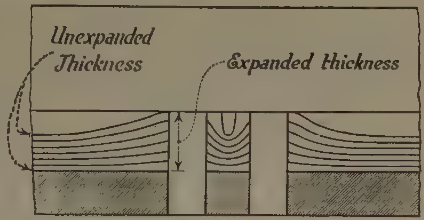


Fig. 21.—Must lift roof without flexing or must lift it as in Fig. 22.

An effort has been made in Fig. 22 to show what would happen if the lower part of the roof expanded x inches and the pillars did not crush or sink in the floor under the load. The assumption is made that the expansions over pillars B, C and D and on the interior edges of pillars A and E are equal. Then the points above A and E, namely A' and E', would be fulcrums of distortion on which the rock above would be bent. Provided there were no weight to the roof, the curve assumed would be $A'F'E'$. However, as the roof has considerable weight, it might be that at the center

of the partly mined area, namely at O', the curve would be about at the point where the expansion of x inches would place it. Then the curve between A' and E' will be $A'C'E'$, a broken-backed anticline.

As the pressure would be relieved on B, C and D pillars it is readily seen how large a load would fall on the edges of A and E. It might easily be twice what it would be where there is no expansion. It is as difficult to loop a steel spring upward as downward. It is as difficult to cause a roof to hog as to sag. The strains resulting from expansion must be clearly apprehended or some of the conclusions later drawn may be held improbable. As the immediate roof over room pillars probably rarely expands to such an extent as to raise the surface, the illustration merely affords a suggestion of the stresses that will come upon props and timber sets which attempt to hold up the immediate roof during the expansion from exposure to the mine air.

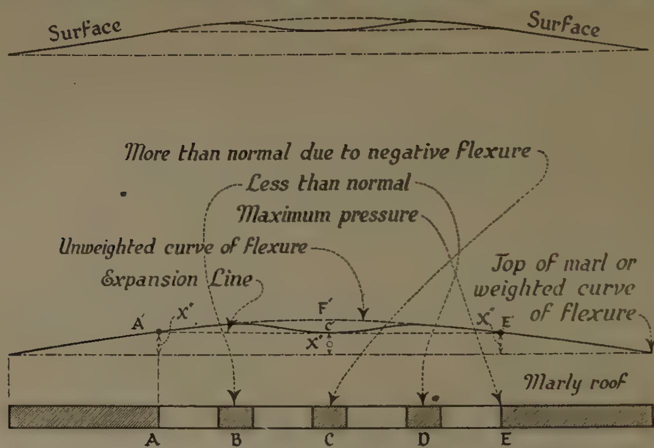


FIG. 22.—ASSUMING PILLARS DO NOT CRUSH OR SINK IN FLOOR, BEAM IS HOGGED AT EDGE OF BARRIER PILLAR.

It is easy to understand that timber sets and props will be demolished by the load brought to bear on them unless certain precautions are taken to reduce that load, and not even then if the expansion is extreme. It is well known that props do not hold up the roof but merely the rock which draws from the roof for any one of several reasons or under any one of several occasions into which it is not well here and now to enter. The rock that draws from the roof may be a few inches to perhaps 100 ft. thick depending on the circumstances which cause the separation of the two. The draw slate may be compared to the plaster on the ceiling of a hall or room. The props are intended to hold, as it were, the plaster and not the ceiling itself.

If, however, the main roof comes down so far that the flexibility of the prop, or its compressibility or the compressibility or extrudability of its base, cannot provide the prop with an opportunity to "beat a well-organized retreat," so to speak, before the overwhelming load the whole weight falls on the prop and it is destroyed. That is sure to happen eventually to any prop left in place where the pillars are removed, but it will not in first mining occur so rapidly and surely with inexpansible roof as it will where the roof expands. It will happen then even if the place is only 8 or 10 ft. wide. The prop which is able to uphold only a few tons or feet of rock is called upon to support the entire roof above it and even to aid in flexing the roof upward, at least in places.

That is one reason for setting loose props, for placing pointed props that will penetrate the floor or will broom under pressure or do both, for omitting floor boards

on soft clay bottoms and for introducing, in some cases, props with sliding members, wedges and compressible and expressible materials which props will shorten when any weight greater than they are intended to sustain is laid upon them.

Just what action should be taken to protect a room against the effect of roof expansion will have to be determined in any case. A tight prop saves the roof from being exposed to the air except surficially and so prevents blistering, because any post will hold up a thin layer that tries to sag. On the other hand, a badly swelling roof will sag between even close supports, and the props will not save it from breakage. Consequently the expansion over the prop will be excessive.

Apparently at the Nova Scotia mines described by Mr. Hay, the lime in the marly roof has been more or less leached near the exposed edges of the coal field just as it has been leached near the outcrop from the ironstone in the Birmingham iron ore fields. Thus the roof under deeper cover, because it contains lime or at least more lime than it does near the surface, is more treacherous than under shallower cover which is a somewhat unusual condition. A similar difficulty occurs with the floor.

However, Mr. Hay does not chance thus to describe it but merely says[1] "Coincident with an increase in cover the roof shales became weak and in some cases the floor, which varies from fireclay to shale, also weakened, and serious difficulty was experienced in keeping the chambers open. Even in narrow-work development places it was necessary to support the roof with steel rails weighing 85 lb. per yard, used as booms and set at 1-ft. centers."

That the weakness is not inherent but is due to an expansive rock is suggested by the sentence following: "Sometimes relief was got by driving an additional narrow place parallel to and slightly in advance of the main road. This place was allowed to collapse every 80 ft. or thereabouts and appeared to relieve the pressure somewhat on adjacent roads."

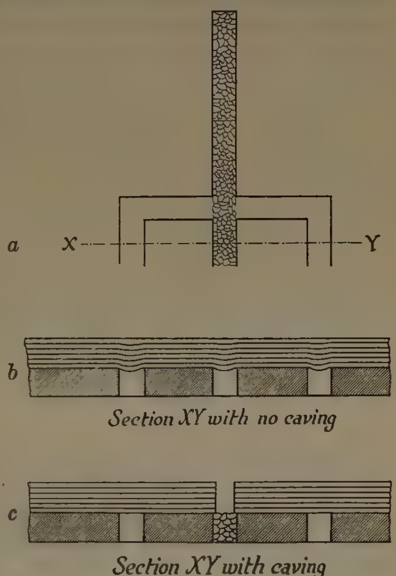


Fig. 23.—Suggesting way in which sagging is prevented at Nova Scotia mines.

The fact that work on an adjacent road prevented rock falls in the other roads suggests that the rock was not inherently weak, and Fig. 23 has been drawn to show the actual situation, at least as it commends itself to me. Fig. 23*a* shows such an entry and Fig. 23*b* a cross-section with all three headings standing open. The expansion has caused a sagging of the roof. A sag may arise from vertical or lateral expansion, but in the first instance it probably comes largely from the latter because only the exposed surface is at first affected. Suppose the skin affected is 3 in. thick, the vertical expansion if 2 per cent., will be only 3×0.02 in. or 0.06 in., about $\frac{1}{16}$ in., and the lateral expansion will be over 120 in. in a 10-ft. entry and will be 120×0.02 or 2.4 in. Such an expansion would cause the rock layer if it drooped in a vertical arc to descend 1 ft. in the center of the airway. A layer thus bent would probably break and fall. It would not bring much pressure to bear if kept in its original position by a crossbeam but if not well sustained it would expand a little, lower a

[1] See page 50.

little, crevice a little, expand some more, lower still more, crevice again and finally fall. In doing so it would open up the mass to further lateral expansion and further crevicing with an increase of vertical expansion. Against this action, early, close and effective support would be valuable. The 85-lb. rail, which is set doubtless sometime after the coal is extracted, performs this needed service.

But the advance place solved the problem in another way. It provided a space into which the laterally expanded immediate roof could extend as shown in Fig. 23c, provided a "smooth" or "smooths" existed that would permit of such motion. After such expansion had thus expended itself, there was no reason for any vertical expansion, for such an effect could arise only from the breaking down of the rock from hindered lateral expansion and that hindering was prevented by the artifice described. Of course, to drive two places to get one, and three places to get two, would prove annoying as would also the delays in getting the advance place ahead so that it would be caved before the other places come abreast of it.

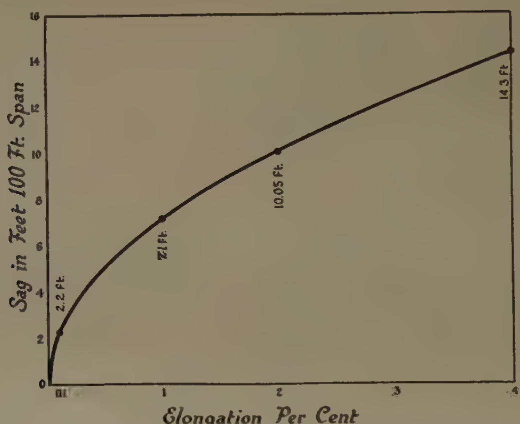


FIG. 24.—SAG FOR ANY GIVEN PERCENTAGE ELONGATION.

Mr. Hay says that, at Princess colliery, rooms could not be driven satisfactorily even a length of 200 ft. "Frequently the roof would collapse, shearing at the ribs, or the floor would give way and heave, almost entirely closing the place overnight." These are natural results of vertical expansion. The expanded material shears itself free of the unexpanded. When it expands over the pillar it bends outward, just as the rock over the opening bends downward, and in consequence the rock over the pillar edges may fall out into the heading.

"Ribbing in," says Mr. Hay—"skipping the pillar" we would term it—was necessary at considerable cost and even then a large percentage of the coal was lost. "Under these conditions it was necessary to change the system of mining and a start was made on the longwall method in 1923."

Why is longwall advancing the preferred method? It is true that it has been found the more successful system. It is probably true in fact, as J. F. K. Brown remarked recently, that it was the salvation of the situation in Nova Scotia. But why? This seems to be the explanation. The rooms which were subject to falls are now replaced by gateways which are surrounded by fallen goaf into which the roof above the gateways can expand laterally. So much for lateral expansion. Vertical expansion also produces less trouble than it would with ribs which resist compression. The packwalls on either side give way sufficiently to enable the rock over them to

expand in step with the rock over the roadway. The roadways also are fewer and so can be better watched and less expensively protected.

By advancing rapidly, new roof can be provided constantly at the face. Under this the men can work in safety. The rigidly timbered roof near the face wall will not expand unduly until the face has moved forward and the timbers have been removed. The roof having broken, toward the gob, expansion in that direction is possible. Only vertical expansion can make trouble and that is roughly proportional to the time of exposure which in turn is proportional to the distance from the face so that the expansion can take place without strain.

I have prepared some figures illustrating the sag due to certain expansions (see also Fig. 24):

Expansion, Per Cent.	Ratio of Expansion.	Sag for 10-ft. Opening, In.	Sag for 100-ft. Opening, Ft.
0.001	0.00001	0.5	4.6*
0.01	0.0001	0.8	8.4*
0.02	0.0002	1.2	12.1*
0.03	0.0003	1.5	1.2
0.04	0.0004	1.7	1.4
0.05	0.0005	1.9	1.6
0.06	0.0006	2.1	1.7
0.07	0.0007	2.3	1.9
0 08	0.0008	2.4	2.0
0.09	0.0009	2.6	2.1
0.10	0.001	2.7	2.2
1.00	0.01	8.5	7.1
2.00	0.02	12.1	10.05
3.00	0.03		
4.00	0.04	17.1	14.3

* Inches.

The smallest of the expansions listed, 0.001 per cent., is about that which would be obtained by raising the temperature of rock 1° C., for Prof. W. J. M. Rankine in "A Manual of Civil Engineering," quotes Mr. Adie as saying that sandstone expands 0.0009 to 0.0012 per cent. and slate 0.00104 per cent. between 0 and 100° or an average of about 0.001 per cent. for 100° and 0.00001 per cent. for 1°. When a place is first excavated in the bed the introduction of air conceivably might change the temperature of the rock exposed about 10°. This would bring quite a strain on the rock surface as the estimate of sag, namely 0.8 in. in a 10-ft. opening, suggests. In some cases perhaps the expansion of the rock from chemical action may offset the expansion from change in temperature and in others the two may act conjointly, to increase the expansion. Attempts have rarely been made in this country to measure either the natural roof temperatures or the expansion from the physical or chemical effects of wetting or air slaking. The figures show how important the effects of a slight expansion or contraction are. They also show that the flexing is not proportional to the expansion or contraction but is greater for smaller expansions or contractions. By increasing the expansion from 0.001 to 4 per cent., or 4000 times, the sag is increased from about 0.5 to about 17.1 in., or only about 34 times, and if the table were carried further the result would be even more marked. The flexing is great with extremely small percentages of expansion or contraction.

Barrier Pillar Legislation in Pennsylvania

By George H. Ashley,* Harrisburg, Pa.

(New York Meeting, February, 1930)

The Legislature of the Commonwealth of Pennsylvania at its last session passed a new act dealing with barrier pillars, which may have a wide interest in other states. In the past the laws of Pennsylvania have required that the superintendent of a mine "shall not permit the mining of coal within 50 ft. of any abandoned mine containing a dangerous accumulation of water, until said danger has been removed by driving a passageway to tap and drain off said water, as provided for in this act: Provided, That the thickness of the barrier pillars shall be greater and shall be in proportion of one foot of pillar thickness to each one and one-quarter feet of water head, if in the judgment of the engineer of the property and that of the district inspector it is necessary for the safety of the persons working in the mine."

Thus it has happened that a thickness of several hundred feet has necessarily been left in pillars in mining. It has seemed to many in the mining industry that this left an unnecessary amount of coal in pillars, and did not provide that the pillar be left equally on the two sides of the mine. As a result the General Assembly of 1927 by Joint Resolution authorized the Governor to appoint a Commission of "seven prominent citizens, of whom four shall be recognized mining engineers, one shall be a hydraulic engineer, one a geologist of recognized standing in his profession, and one a State mine inspector from the bituminous coal mining district, to investigate and report upon barrier pillars in the bituminous coal region." The Governor appointed the following Commission: Morris Coulter, Clearfield Bituminous Coal Corp., Indiana, Pa.; A. E. Roberts, Monroe Coal Mining Co., Revloc, Pa.; George T. Lewis, Westmoreland Coal Co., Irwin, Pa.; Morris Knowles, hydraulic engineer, Pittsburgh, Pa.; Thomas A. Mather, State Inspector of Mines, Tyrone, Pa.; A. B. Jessup, Anthracite Coal Operators Association, Jeddo, Pa.; George H. Ashley, State Geologist, Harrisburg, Pa. Mr. Coulter was elected Chairman and Mr. Ashley, Secretary.

This Commission held meetings in Harrisburg, Johnstown and Pittsburgh. It obtained and considered the laws and experiences of other states and European countries. Some of its members conferred at length with the directors and other members of the U. S. Bureau

* Pennsylvania State Geologist; Secretary Barrier Pillar Commission, Department of Internal Affairs.

of Mines and U. S. Bureau of Standards, and the Commission had and used the results of tests on coal and other studies carried on by those bureaus. Members of the Commission attended a symposium on the subject of barrier pillars especially set up for the Commission by the American Institute of Mining and Metallurgical Engineers and a similar meeting held under the auspices of that Institute on ground movement and subsidence. The minutes of these meetings were graciously put in the hands of the Commission. From many sources came data dealing with the strength of coal, experience with barrier pillars and much other desirable and necessary information.

After preliminary discussions, the Commission prepared a questionnaire which was sent to all of the bituminous mining companies of the State, and received in return a large amount of information and recital of experience. Later two largely attended all-day public hearings were held, one in Pittsburgh and one in Johnstown. At the request of the Commission George S. Rice, Chief Mining Engineer, U. S. Bureau of Mines, during a European trip made a special study and inquiry of experience in England and elsewhere.

As a result of a study of all the information thus accumulated, the following general conclusions were reached:

1. The Commission found no record of a single instance in which water accumulated in one mine has broken through or pushed out an undisturbed barrier pillar with fatal results to men in lower workings, even though in some instances subsequent mining revealed that at points the pillar had been reduced to 10 ft. or less.

A report covering 84 drowning disasters in England showed that all but 18 were due to mining through into flooded workings where mine maps of those workings were faulty or lacking. The 18 were cases of running into water-bearing strata over denuded coal, or into faulty and dislocated strata. U. S. Bureau of Mines *Bulletin* 115 lists seven drowning catastrophies out of 308 major accidents. Three of these are listed as due to the inrush of surface waters or sand; of the four remaining, one was due to inrush of water from an old shaft. The other three were in Pennsylvania, and on investigation by the Commission, it was found that in each instance mining had been carried into forgotten or poorly mapped workings, "Through," as one of the juries found, "misleading maps and drafts."

2. Experience shows that although thickness of pillar in some instances has an effect on the amount of seepage or flow of water through the strata, in general it is not possible to prevent such flow by specifying a minimum thickness of pillar, as water has been known to flow an indefinite distance through the strata below and above the coal, and even through the jointing of the coal bed itself. Because of this, effort was concentrated on determining the minimum thickness of barrier pillar necessary to main-

tain itself indefinitely under the conditlons of bed section, character of coal, depth of cover, character of roof and floor existing at any place.

3. At the Pittsburgh hearing it seemed to be the consensus of opinion that a pillar 100 ft. in thicknesss is sufficient to maintain itself against any head of water, for any bed of coal up to 10 ft. thick, and that for thinner beds, or moderate heads of water, 50 ft. is ample, provided the integrity of the pillar can be insured. At Johnstown, opinion, though not so unanimously expressed, trended in the same direction and toward the same figures.

4. The data obtained and general experience indicated that to maintain intact at least a portion of the pillar would require a thicker pillar for a thick bed than for a thin bed, other things being equal. Also a thicker pillar would be required to maintain itself under heavy cover than under light cover.

5. By a convergence of the evidence the Commission reached certain conclusions as to a safe minimum pillar for selected thicknesses of coal under selected depths of cover, assuming a large factor of safety to cover the less favorable conditions.

6. Having reached these figures a rule or formula was devised that, as closely as possible, just covered these determined minimum requirements. Briefly, it is that the minimum pillar shall be not less than 20 ft., plus four times the thickness of the coal bed, plus 10 ft. for each 100 ft. or fraction thereof of cover at the boundary in question.

7. Table 1 shows the application of this formula to various thicknesses of coal and depths of cover, it being understood that the act provides for one-half of the pillar to be left on each side of the property boundary line, save only where prevented by mining done prior to the approval of this act.

TABLE 1.—*Proposed Thickness of Barrier Pillars*

Thickness of Bed, Ft.	Thickness of Cover, Ft.									
	100	200	300	400	500	600	700	800	900	1000
1	34	44	54	64	74	84	94	104	114	124
2	38	48	58	68	78	88	98	108	118	128
3	42	52	62	72	82	92	102	112	122	132
4	46	56	66	76	86	96	106	116	126	136
5	50	60	70	80	90	100	110	120	130	140
6	54	64	74	84	94	104	114	124	134	144
7	58	68	78	88	98	108	118	128	138	148
8	62	72	82	92	102	112	122	132	142	152
9	66	76	86	96	106	116	126	136	146	156
10	70	80	90	100		120	130	140	150	160

8. It is recognized that in areas of crushed or faulty coal, or where coal is subject to squeezing, it is not possible to prescribe a general rule

that will apply to all cases without making the rule burdensome under normal conditions. The proposed act, therefore, provides for the exceptional case as well as the normal.

9. The Commission also wishes to call attention to the figure 12, in Section 17 of Article IV, same act, and to recommend that that figure be increased to 20. The figure refers to the distance to which drilling must be maintained ahead of workings designed to tap water in abandoned workings. Under present-day mining methods, using up to 9-ft. undercuts, this may leave only just over 3 ft. of coal, which might conceivably blow out into the flooded mine. It is believed that the proposed figure of 20 ft. will greatly increase the safety of tapping old workings.

On the basis of these findings the Commission prepared a proposed act. After the printing and distribution of this act members of the mining fraternity of western Pennsylvania asked for additional hearings, and two such additional hearings were held leading to some modifications of the act as originally proposed.

The act as passed, is here given in full. Words in brackets are words or clauses omitted from previous acts.

No. 190

AN ACT

To amend section five of article three, and section seventeen of article four, of the act, approved the ninth day of June, one thousand nine hundred and eleven (Pamphlet Laws, seven hundred fifty-six), entitled "An act to provide for the health and safety of persons employed in and about the bituminous coal-mines of Pennsylvania, and for the protection and preservation of property connected therewith," further regulating barrier pillars on certain property lines and providing procedure for the settlement of disputes.

Section 1. Be it enacted, &c., That section five of article three of the act, approved the ninth day of June, one thousand nine hundred and eleven (Pamphlet Laws, seven hundred fifty-six), entitled "An act to provide for the health and safety of persons employed in and about the bituminous coal-mines of Pennsylvania, and for the protection and preservation of property connected therewith," is hereby amended to read as follows:

Section 5. The superintendent shall not permit the mining of coal within fifty feet of any abandoned mine or an abandoned portion of any mine containing a dangerous accumulation of water, until said danger has been removed by driving a passageway to tap and drain off said water, as provided for in this act. [Provided, That the thickness of the barrier pillars shall be greater and shall be in proportion of one foot of pillar thickness to each one and one-quarter feet of water head, if in the judgment of the engineer of the property and that of the district inspector it is necessary for the safety of persons working in the mine.] *The superintendent shall not permit the mining of coal in any seam the entire distance to a property boundary line (not including boundaries around reservations or along crop lines), when, on the adjoining property there are mine workings in said seam within three thousand feet of said boundary line, but shall leave a barrier pillar, from the operation to the property boundary line, of not less than ten feet plus two feet for every foot or part of a foot of thickness of the bed measured from the roof to the floor, plus five feet for each one hundred feet or part of one hundred feet of cover over the bed at the boundary line; and, where the coal on one side of the property boundary line shall have been mined prior to the passage of this act closer to the property boundary*

line than hereinbefore permitted, then the barrier pillar to be left in the mine approaching the boundary line shall be at least equal (when added to that already left in the adjoining mine) to that hereinbefore required on both sides of said property boundary line: Provided, That, if in the opinion of the district mine inspector or the superintendent of either mining property, the barrier pillar, as hereinbefore required, is deemed insufficient, then, after due notice to the operator or operators of the mining property adjoining, a barrier pillar of unmined coal (one-half of which shall be on each side of the property boundary line, except as provided above in this section) shall be left, of such thickness as in the judgment of the district mine inspector and the superintendent or owner of either mining property is deemed necessary to afford safety and protection: And provided further, That, if it shall be agreed by the superintendents of such adjoining coal mining properties that such property boundary line is so located that there is no danger to property or lives in mining coal on either or both sides of the property boundary line up to said property boundary line, then in such cases mining to the property boundary line shall be lawful, if all danger from accumulated water and gas shall have first been removed by driving a passageway to tap and drain off any accumulations of water and gas, as provided for in this act: Provided, That if any of the parties in interest fail to agree on the carrying out of any of the provisions of this act, any one of said parties may appeal to the Secretary of Mines, who shall appoint three reputable and competent mining engineers, who shall constitute a commission whose duty it shall be to investigate the conditions and determine the necessity for leaving barrier pillars, or a barrier pillar of greater thickness than is provided for in this act, or the adequacy of an existing pillar to withstand a water head, as the case may be, and the decision of said commission, or a majority of its members, shall be final and conclusive, and binding on all parties concerned. The members of the said commission shall be compensated for their services, by the property owners, in an amount to be fixed by the Secretary of Mines. Before any such appeal to the Secretary of Mines is taken, notice thereof, in writing, shall be served personally or by registered mail on the opposite party, setting forth the date when such appeal will be taken, and proof of the service of such notice shall be filed with the appeal.

Section 2. That section seventeen of article four of said act is hereby amended to read as follows:

Section 17. In any working place that is being driven within supposedly dangerous proximity to an abandoned mine, or portion of an abandoned mine, suspected of containing explosive gas, or that may contain a dangerous accumulation of water the mine foreman shall see that at least two bore holes shall be maintained not less than [twelve] *twenty* feet in advance of the face, and, on each side of such working place, bore holes of the same depth shall be drilled diagonally, not more than eight feet apart, and any place driven to tap water or gas shall not be more than [eight] *ten* feet wide. No water or gas from an abandoned mine, or portions of an abandoned mine, and no bore hole from the surface, shall be tapped until the employes, except those engaged at such work, are out of the mine, and such work shall be done under the immediate instruction and direction of the mine foreman, with the use of locked safety lamps.

APPROVED—The 10th day of April, A. D. 1929.

JOHN S. FISHER

The foregoing is a true and correct copy of Act of the General Assembly No. 190.

(Signed) Charles Johnson
Secretary of the Commonwealth

DISCUSSION

Discussion at Annual Meeting, February, 1928

G. S. RICE, Washington, D. C.—This paper deals with a question of great importance to coal-mining men, affecting, as it does, not only the safety of miners but also

the coal reserves, and it therefore deserves very careful consideration. Pennsylvania leads all of the states in the annual production of coal and has done so from the beginning of coal mining in this country. It is therefore natural that it has been a pioneer in coal mine legislation in the United States. Hence it was deemed advisable by the Institute committee that the problem of barrier pillars in coal mines brought forward by the recent legislation in Pennsylvania should be fully discussed to determine to what extent the provisions of the act may be applicable in coal mines in other states.

As Dr. Ashley has mentioned in his paper, the general subject was discussed by this committee in February, 1928, following an informal presentation of the matter by him, and some of the points brought forward at that meeting, by coal-mining men of wide experience, were considered by the committee as worthy of preservation. This discussion of 1928 follows:

G. H. Ashley.—The bituminious act (that effective prior to 1929) provided, in Art. III, Sec. 5, that a barrier pillar of not less than 50 ft. should be left and that where it was below water level it should have a thickness of not less than 4 ft. for every 5 ft. or that equivalent, 1 for every 1¼ ft. of water head unless the engineer of the company concerned and the mine inspector for that district should agree upon some modification of that rule. For example, a 400-ft. pillar would be required for a depth of 500 feet.

The agreement relating to pillars between the properties of certain companies, made by the engineers of several of the anthracite companies, with the state mine inspector of that district, provides that the thickness of barrier pillars shall be five times the thickness of the bed at water level, and below water level, plus one per cent. of the water head. For a bed only 3 ft. thick, which is common in the central Pennsylvania bituminous field—3 × 5 is 15, and under a 500-ft. head it would be 5 ft. more—a 20-ft. pillar would serve, whereas under the bituminous rules at present (Feb., 1929) a 400-ft. pillar would be required.

It is that sort of thing which has led so many people to think that our present rule is excessive in its demands. Furthermore, the bituminous rule does not require that the pillar be equally divided between adjoining properties. It only provides that a property being mined up to another property in which there is accumulated water, must not leave a pillar less than the provision made. The anthracite law, on the other hand, does provide that the barrier pillar shall be agreed upon between the chief engineers of the two properties in conjunction with the mine inspector of that district, so it is there evenly divided.

T. A. Mather, Tyrone, Pa.—The Pennsylvania bituminous act (prior to 1929) was based on the theory that we should leave an equal weight of the coal to equalize the head of water. As you are aware, water weighs 62½ lb. per cu. ft. and coal about 79 to 80 lb. per cu. ft.; accordingly by weight it requires 1¼ cu. ft. of water to equal 1 cu. ft. of coal. So, the theory was that a water head in excess of this ratio would be able to move a block of coal that was assumed to be free to lift or slide.

G. H. Ashley.—We have been taking the advice of Mr. Knowles, a hydraulic engineer, who is a member of the Commission, that a barrier pillar under the condition in which it exists in the mine does not serve like a gravity dam in the sense of a hydraulic dam where weight against water is the determining factor, but as a beam from roof to floor because it is weighted on top. The weight of rock exceeds by 2½ times the weight of water, so it becomes a question of internal resistance to frictional sliding or the strength of coal as a beam to resist shearing stress applied parallel to the bedding.

The feeling we had as we discussed the matter was that in the solid bed we are concerned with free partings or soft material softened by the water or something of that kind, and the hydraulic strength required of a barrier pillar from the standpoint of withstanding a water head would be small as compared with the strength of pillar

needed to maintain its integrity under load, when the adjacent coal is mined out and as the roof settles. So the problem appears to be largely one of having a barrier pillar which will stand the spalling and roof action that may take place during or following mining along its side and still give sufficient strength to maintain itself indefinitely.

W. E. Fohl, Pittsburgh, Pa.—So far as you have gone, then, your thought is that if the pillar is able to support the roof, it is more than able to withstand the water pressure?

G. H. Ashley.—Yes. As I said, that perhaps needs checking up on the figures but that is the impression we had.

H. G. Moulton, New York, N. Y.—I should think that the committee might agree that the smaller you can make barrier pillars and have them safe, the better we can approve them as a committee, in order that the coal may be conserved.

G. H. Ashley.—If you figure on a 48-acre tract, with a 400-ft. pillar under a 500-ft. head, it means you leave a third of the coal; if you have a 160-acre tract, you would leave a sixth of the coal. That is the loss you are trying to avoid.

G. S. Rice.—In western Pennsylvania do you not have coal in Greene County which is much deeper than 500 ft., and which would therefore cause still larger losses of coal reserves in barrier pillars?

G. H. Ashley.—Yes, the Pittsburgh bed is 1,000 ft. below drainage besides the additional height of the hills.

G. S. Rice.—Is it not probable that still lower beds may be found that under future market conditions may be workable?

G. H. Ashley —There are the Lower and Upper Freeport coals which we know little about, but oil and gas drilling indicates 6 ft. of material which is called "coal," It may be or may not be workable. This is 600 ft. below the Pittsburgh bed. Other beds below that may prove to be workable, although the drill records do not usually indicate workable coal.

G. S. Rice.—You perhaps might have as much as a 1000-ft. head of water in the future?

G. H. Ashley.—Yes, we may have 1000 to 2000 in the future.

H. G. Moulton.—Is the Commission giving consideration to the inclusion of a rather complete collection of all known failures of pillars?

G. H. Ashley.—That is one of the things we are peculiarly interested in getting. Mr. Thomas, of the State Mine Inspectors, has made the statement that so far as we know there never has been a failure of a barrier pillar because of thinness. It may leak but it has never failed in the sense of giving way.

G. S. Rice.—May I ask, if you have a failure in an anthracite pillar, such as a member of this committee informally indicated has occurred, whether it would be a fair presumption that a bituminous pillar under the same conditions would fail?

G. H. Ashley.—As I understood it, that was not a failure in the sense of the barrier giving way.

A. B. Jessup, Jeddo, Pa.—That is true but leakage is total. It is a complete success as far as leakage is concerned.

G. H. Ashley.—I want to ask in regard to that whether, from the situation as described, one might be led to infer, that thickness of pillar had little to do with the leakage; that the pillar in question would have leaked if it had been twice as thick.

A. B. Jessup.—The leakage is apparently through the strata immediately over the seam of coal and not through the coal itself. The barrier is 200 ft. thick at that point, the seam 14 ft. thick and the head not more than 200 ft. It is in a V-shaped

basin. The coal pillar is far too thick to be crushed by the weight of cover or to give way under the head of water and it does not leak, but this mass of solid coal is utterly ineffective in preventing inflow of water. The water runs between layers of roof slate and drops down through jointings several hundred feet beyond the barrier.

T. A. Mather.—I do not think the thickness of a barrier has any bearing whatever on the amount of leakage that may have occurred through the pillar or through the overlying and underlying strata. There were in the anthracite district a number of cases where there was over 900 ft. laterally of coal but leakage of water occurred.

G. H. Ashley.—Geologically we have come to look upon coal as not unlike sandstone as a water carrier because it is usually underlain by a clay that will not permit movement of water. In field work we constantly trace the outcrop of coal beds by the springs if we are working on the down-dip outcrop of the coal, indicating how universally it is a water carrier. In that case, of course, it may be unmined coal, so that the thickness away from the crop has nothing whatever to do with it.

G. S. Rice.—I suppose the Commission, while not hoping to make water-tight barrier pillars, does give consideration to a strength of pillar that will prevent the danger of sudden rushes and flooding. Such cases in coal mines fortunately have been rare in this country, but metal mines in the North have experienced disastrous rushes of water from glacial drift. There have been some disasters of this kind in English mines through mining too close to areas which have been filled with water. The only record that I recall of a serious disaster by flooding in the bituminous mines in this country was in Northern Illinois where a break of shallow surface, not a pillar, let in a flooded stream which drowned 69 men; that was in 1883. In the anthracite district, there was, I believe, an inrush of quicksand at Nanticoke, in 1885, in which 26 men were lost, but I understand this also was not a case of a pillar giving way.

G. H. Ashley.—Of course, the primary thought is this question of safety to life rather perhaps than the economic side of pumping water from another's property.

The bituminous field in Pennsylvania differs from the anthracite where there are water-bearing sands, as in the Wyoming Valley, and from northern Illinois and other fields where the drift has considerable depth. In the bituminous fields of Pennsylvania, there are considerable depths of sands only in the valleys of the Allegheny and Ohio rivers—not over perhaps 25 ft. in the Allegheny but reaching to a depth of 150 to 180 ft. possibly in the Ohio. But those cases, unless we set our barrier pillars under the river itself, would hardly come under the general problem of safety in mining. I think our task is not to guard against the surface cave-in, but to consider only the strength of the barrier pillars within the coal bed, which will prevent danger to life from the giving way under water head and this concerns the integrity of the pillar itself to resist crush.

G. S. Rice.—When Dr. Ashley and other members of the Commission visited the Bureau of Mines and other bureaus soon after their appointment, one of the problems discussed was whether physical tests of the strength of coal should be made, considering it not only as a pillar supporting a heavy overburden, but also as a beam extending from floor to roof to resist the hydraulic pressure. It was agreed the former was the critical question, but how was the compressive strength to be tested? By small cubes or columns in a compressive testing machine and if not, by what method?

I then expressed views based on observing and studying many tests of specimens, in cubical form or columnar form, beginning with testing carried on for the Pennsylvania Anthracite Mine Cave Commission in 1913, jointly by the Bureau of Mines and Bureau of Standards at Pittsburgh,[1] that the test data thus obtained do not provide proper criteria for the strength of mine pillars.

[1] Tests of Strength of Roof Supports Used in Anthracite Mines of Pennsylvania. U. S. Bur. Mines *Bull.* 303 (April, 1928; printed 1929).

Although the method is one that has always been used to test the compressive strength of building material, in the course of the testing it become increasingly apparent to me that the method is unsuitable for determining the strength of a mine pillar.

As I stated in a Review of the Compressive Strength of Anthracite, Bituminous Coals and Mine Supports, contained in an explanatory introduction in *Bulletin* 303, "If a coal pillar is left of sufficient size, so that the spalling or crushing of the edges will not affect the interior, the average unit strength will be far higher than that developed in a compressive machine by a specimen that is free to rupture on four sides."

On the other hand, if a coal bed contains a clay parting, or the bed is overlain or underlain with a plastic clay, when in a mine pillar under heavy load, the clay flows slowly toward the free faces of the pillar. If the coal bed has faces or vertical joints as most coals do, the movement of the clay tends to force off slabs from the faces of the pillars.

These conditions cannot be duplicated by relatively small blocks of coal in a compression machine. Ift he blocks are small, they do not contain the natural joint planes; on the other hand, it is impossible to cut out large blocks, remove them from the mine and place them in the machine without some opening of the joints. These troubles were experienced in the U. S. Bureau of Mines compressive tests of Pittsburgh coal.

These views evidently coincided with those of the Commission, and the proposal of testing the strength of coal under compression was dropped.

There was some discussion about making tests at the Bruceton Experimental Mine of the strength of pillars as a dam, the proposal being to create artifical water heads by pumping until the respective test pillar gave way. The funds not being available, the matter had to be dropped, but I was requested to investigate the status of barrier pillars in European mines, incidental to a visit I had been detailed to make.

The results of the inquiry in Europe was as follows: In Great Britain, questions of the size of barrier and boundary pillars have been raised from time to time, especially in connection with preventing water from passing from one mine to an adjacent mine further down the dip of the coal bed. There also has been legislation on questions relating to the size of pillars for support of railways and canals.

So far as boundary pillars are concerned, there is no specific law on the subject, but the general rule is that the owners of the coal in the ground insist when leasing, that a pillar of coal should be left by the lessee along the boundary. On the other hand, where there is no proviso, the colliery may work the coal to the boundary. In the opinion of Prof. Henry Louis, an eminent mining engineer and a member of the Government Commission on the support of railways and canals, if boundary pillars are to be left at all, they must be strong enough to resist permanently superincumbent pressure. This, to his mind, is a question that could not possibly be answered in general terms: the ultimate resistence of a pillar to crush, will depend upon immediate roof and the character of the workings on either side. For example, if the workings on either side of the pillar are closely stowed or packed, a narrower barrier pillar would be required than if the roof were unsupported by packing. Again, a roof that is brittle and breaks off short along the side of the boundary pillar would crush the pillar far less than a strong roof which would hang together on either side the pillar, and thus bring a greater weight upon the pillar.

In the final report of the Coal Conservation Committee of the Ministry of Reconstruction (final report, 1918, page 55), the Committee recommended that the areas of pillars of coal left in bord and pillar workings at different depths from the surface, should be, speaking roughly, proportional to depth. This, however, does not have direct application to the question of the boundary requirement because in Great Briain, that is a matter of private agreement between lessor and lessee.

It has been customary, however, in Great Britian, in the absence of the specific agreement, to leave a pillar 20 yards in thickness, 10 yards on either side of the boundary.

Following the World War, in the evidence given in discussions before the Sankey Commission (1919), the Federation of Miners pointed out, as argument for the nationalization of the coal industry, including the taking over of coal rights, the wastage of coal left in barrier pillars between different holdings and the lack of economy in having numerous small pumping plants instead of a few highly efficient central plants. However, most mining engineers who gave evidence, very strongly attacked this latter idea and showed that while there might be economies in the use of power in individual large central plants, there would be enormous losses of total power consumed if the water entering near the edges of basins was allowed to flow down, perhaps to a depth of several thousand feet before it was pumped out against this great head.

Some very interesting facts were related in a paper, The Underground Barriers of the Cannock Chase Coalfield, by J. C. Forrest, which, with the discussion, appeared in the May, 1920, issue of the *Transactions* of the Institution of Mining Engineers (Great Britain). In this case, large barrier pillars were employed to prevent large flows of underground water from higher into lower mines of the basin. The conditions were complicated by profound geologic faults. Large barrier pillars were left adjacent to these faults. Mr. Forrest states (p. 117):

"The barrier, to be of any use, should be sufficiently strong in each seam at the one point in a vertical line, and should usually be thicker in the bottom seams than in the top ones. I think a 60-yard barrier at a depth of 250 yards should hold back any water (the bulk of the barriers left are from 20 to 40 yards). But this does not always follow, as I have had to leave barriers much thicker in the upper seams because the workings in the lower seams had caused breaks higher up which it was unsafe to pass in working the upper seams."

Mr. Forrest also says that if a fault crosses a barrier it is necessary to thicken the barrier for some distance on either side, and if there should be a bed of "green rock" to form any part of the barrier, it would be necessary in that case to make the barrier at least four times the usual width. A map illustrating the situation shows barriers at different points in the coalfield, varying in thickness from 60 to 100 or more yards.

Mr. Forrest sums up admirably the advantages of barriers used under the conditions, which represent a loss of only 1½ per cent. of the coal area, as follows:

"1. Dry pits, with untold comfort to workmen and to the animals employed.

"2. Great saving in the cost of working the mines.

"3. No necessity to raise as much water as coal.

"4. Saving of at least 25 per cent. of the coal area that has been and is being worked." By No. 4 is meant that some of the area would have been impossible to work because of the large amount of water which would be encountered.

In the discussion which follows, Jonathan Hunter speaks of one case of a barrier pillar against a fault on the rise side which had to be left 200 yards or more in thickness. On the other hand, T. A. O'Donahue (p. 126) said he knew of one case where water with a head of 500 yards is being held back by a barrier only 30 yards wide. He adds that the 30-yard barrier has so far been successful in holding back the water for many years from a group of mines, but he thinks "most of us will admit it is a very narrow margin of safety."

The general feeling of prominent British mining engineers with whom I have talked on the subject, is, that it is impossible to set down fixed figures for the proper thickness of barriers without knowing all of the conditions, *viz.*, pitch of the coal bed, the thickness of the coal bed, whether there are other beds above or below, which have been mined or are to be mined, the method of mining and whether the workings

are stowed or packed, the hydraulic head to which the barriers might be subjected and whether there are geologic faults, and the nature of these faults.

France.—There is no private ownership in France of the coal in place, it is practically leased by the Government; that is, concessions are given for an annual rental and tax on the net proceeds, and as the Government exercises close supervision as to how and where the coal is to be mined, each case of a barrier pillar is, I understand, considered on its own merit.

Belgium.—In Belgium where the coal beds are so intersected by faults, and so folded that the dips are steep that it would be impossible to lay down any hard and fast dimensions for boundary pillars, the safety supervision by the Government is very close.

Germany.—In Germany, most of the coal resources are under private ownership as in Great Britain and in the older coal fields in Belgium. There are mining district safety regulations regarding minimum thickness of boundary pillars. In the Ruhr district, for example, boundary pillars are made 20 m. thick on either side of the boundary. In other words, in the absence of special arrangements, boundary pillars are 40 m. (132 ft.) thick. But, where there are questions of safety involved, as in the case of water held back by a boundary barrier, then it would be the subject of special action by the Chief Mine Inspector of the district.

It may be pointed out that the coal fields of Europe are much more broken by faults, and there are usually many more workable seams in a basin than in the average coal field of the United States. Our coal reserves, while enormously greater than those of Europe, are more spread out, and with exceptions not concentrated in small areas. This makes the problem in European countries, so far as boundary pillars are concerned, a much more complicated matter than in the average coal field of the United States where the coal beds are approximately level.

In the report of the (British) Minister of Reconstruction on the acquisition and valuation of land it is pointed out that if only a small percentage of the barrier coal could be worked there would be a great saving. The conditions under which barrier pillars were left are classified as (1) against danger of water, gas or underground fires, and, (2) to mark the boundary of the coal ownership, but the report indicates that it was mainly in the case of water barriers that the special problem arose. It recommends a survey of all barriers with a view to removal if unnecessary barriers were found and that an assurance fund be established to take care of the conditions of failures.

In the evidence given before the Royal Commission on the Coal Industry (1925), different witnesses gave varying points of view on the need of barrier pillars. Sir Richard Redmayne, former Chief Inspector of Mines, commented that barriers are an arrangement made by royalty owners to safeguard their area and that barriers 22 to 60 yards in thickness are left between respective properties. Under some conditions barriers can be removed by suitable arrangements for ventilation and drainage.

Dr. J. S. Haldane, as President of the Institution of Mining Engineers, presented a general statement from its Council, which pointed out that it is very difficult to express an opinion on the redistribution or relocation of reserves and coal barriers. The Council considered that the use of central pumping stations was misunderstood, that they were valuable in effecting economies where the drainage waters could be assembled at such central plants without increasing the water head, but if the water had to be pumped from a greater depth, due to removal of barriers, the increased head would more than counterbalance any saving from the use of central pumping appliances.

The Duke of Northumberland, an extensive royalty owner, commented that the barriers were left at a thickness of 30 to 40 yards on the boundaries, not merely as boundaries but as an essential part of the layout of the mines upon which drainage

and ventilation systems might depend and as safeguards against risks which might be accrued by adjacent workings in case of mines coming together. He further said, that the total area in the barriers is very large but when a barrier cannot be touched without endangering men, it must, of course, be maintained.

Evan Williams, President of the Mining Association of Great Britain, and W. A. Lee, Secretary, in a long statement regarding the state of the coal industry, said in reference to removal of barriers and of central pumping stations, that it was against the current practice to remove barriers, that the practice was to catch water as near the point of origin as possible. The Mining Association was opposed to the removal of barriers.

A recent British report touching on the question of barriers, mentions the danger of a large volume of water being held at higher levels by barriers, but on the other hand, pointed out that by removal of barriers, that the deep workings naturally dry would be made wet, and moreover in certain cases, there would be the question of pumping water from depths of 2000 ft. or more.

John C. Forrest pointed out that by removal of barriers in the Cannock Chase coal field, many collieries now working dry would be faced with the cost of pumping an extra 2000 tons of water a day. In some cases the barriers hold back feeders of 12,000 tons of water a day, and it is estimated it would cost $1000 daily to pump this amount of water from a depth of 350 yards.

R. D. Hall, New York, N. Y.—Much has been said about the condition of the roof and of the coal but nothing about the condition of the floor. It seems to me that is quite important.

There is a great deal of exudation of the clay from underneath the coal and that has a double effect; it not only ceases to provide the support that would otherwise be obtained but it also thrusts out the coal ribs just as has been described by Mr. Rice in referring to testing the strength of coal. It has been found in England, that when in order to distribute the pressure properly, as they hope to do, they put lead plates underneath the material compressed, the latter does not stand nearly as much pressure as it would when it was on solid material, that the exudation of the lead from underneath the steel weakened the specimen.

Similarly, the clay under the coal in exuding tends to tear off pieces of the coal and to force the clay in between the various faces of the coal and in that way break it up. I think that is quite a consideration because we find the clay rising all around the pillar and there is a possibility also that through that there may be a passage of water underneath the pillar. That passage is probably not one that is dangerous to life, but it might be quite an important economic consideration.

G. S. Rice.—The blocks tested for the Anthracite Mine Cave Commission were very carefully planed on the top and bottom, and were set in plaster of Paris above and below, not using lead. The block was put under slight compression while the plaster was "setting."

H. G. Moulton.—As to the effect of a testing machine on a cube of coal, I think you would get results more comparable with the behavior of coal in a pillar if the specimen tested in the machine had an area of 5 or 6 sq. ft. and a thickness of 1 or 2 in.

It is possible that thin films of clay under slabs of coal having relative dimensions similar to those noted above could not be forced out under pressure. Of course, if it were possible to apply any testing on sheets of coal, even built-up sheets of height and width proportionate to the dimensions of the barrier, the testing machine might give more nearly comparable results.

G. S. Rice.—That subject was discussed at a recent conference and the Bureau of Standards will consider the possibility of making tests of that kind on thin sheets,

perhaps cutting out just one thin bench and testing it to see how it compares with the test in the cube.

As I have frequently stated, I believe compressive tests on specimens meant to represent pillars, should have dimensions proportionate to pillars, which for bituminous coal means widths at least five times the height. However, in the case of coal, I have grown skeptical of being able to cut specimens that are representative of coal in place. Coal is too easily fractured in handling. In reference to the failure of pillars, you will recall that this committee had a very excellent paper by Dr. Crane[2] on ground movement and subsidence effects in iron mines in the Birmingham district. He showed the advantages of leaving loose material against the sides of the pillar in order to get a certain amount of confinement. Eli T. Conner of this committee, pointed out to me and other advisory engineers connected with the Scranton Mine Cave Commission (1911–12) how disastrous has been the effect of pillar robbing along gangways when cleaning up along ribs and taking up some of the bottom coal has given an opportunity for the pillars under heavy load to spall off and start to crush.

A. B. Jessup.—I may say in regard to the particular case of a barrier pillar already discussed, where leakage over the top was so bad, that while we were impressed with the infallibility of the formula for anthracite barriers so far as safety was concerned, yet we did not believe in it 100 per cent. in respect to leakage.

We had had leakage over the top of a barrier 100 ft. thick at the other end of the mine so we made this one 200 ft. thick under similar conditions and complete leakage over the top again occurred. As we retreated from the barrier in drawing pillars we were safe enough from inrushes of water from the inundated mine which might endanger life but we had left 2 or 3 times as much coal behind in barriers as was necessary for safety without even holding the water back. Since we then had the inflow of the two mines to pump we have in mind putting down drill holes from the surface and flushing culm in with the idea of letting the water carry it into the cracks and possibly block them up.

There is one point in connection with barrier pillars which should not be over looked. We consider that one of the values they have is to keep out the gases from possible mine fire on the other side of the pillar. You must also have the pillar thick enough so that the fire will not creep through. For that reason barriers should be of ample size above water level although the formula only gives a nominal thickness of pillar above water level disregarding depth below surface. Intact barriers are also of great value in case you have to flood a mine fire. If you want to flood your own mine, you do not have to ask the other fellow and you are not liable to have an injunction served upon you which will prevent you from doing it. It is also much more feasible and quicker to fill your own mine than it is to fill yours and the adjoining one.

G. S. Rice.—Was there any ground movement caused by mining out in the vicinity of the barrier pillar which leaked so badly—I mean ground movement of the overburden above the barrier?

A. B. Jessup.—No, not above the barrier which was very thick considering it was only 200 ft. deep at the synclinal axis of a V-shaped basin. Water stood against the other side of the barrier in an abandoned mine. When the breast pillars of our mine had been drawn back for several hundred feet from the barrier and the roof had come down the water from the other mine ran over the coal barrier which remained

---

[2] W. R. Crane: Subsidence and Its Relation to Drainage in Red Iron Mines of the Birmingham District, Alabama. *Trans.* (1927) **75**, 837.

W. R. Crane: Roof Support in the Red Iron Ore Mines of the Birmingham District. *Trans.* (1925) **72**, 187.

intact. The coal was solid and impervious to water but the roof rock let several thousand gallons per minute travel through longitudinally and drop into the workings.

It is interesting to note, in regard to the thickness of coal barrier pillars thought necessary for strength, that sometimes when openings in them have been made by accident or made before they were designated as barriers, and these openings have brick or concrete dams erected in them which are only from 7 to 12 ft. in thickness, these relatively thin brick and concrete dams held back very large heads of water as well as the much thicker coal pillars. The hitching into the coal is the vital thing so that it does not give way by leakage through joints around the ends of the dam.

C. Enzian, Fairmont, W. Va.—It has been my privilege, for two or three years, to study the question of barrier pillars quite thoroughly. In that study I might say all the points that have been brought out here have been verified very substantially. The case Mr. Mather spoke about is parallel to one I investigated where there was an area of coal 1000 ft. wide between two adjoining mines. The mine lying on the higher elevation had 60 ft. of water head against the pillar. It became necessary to drive through that pillar to drain the water from it so as to make possible the mining of the coal in the mine lying to the dip side of the pillar. It was found that the water found its way through the overlying strata and partly through the fireclay floor of the seam.

The mine on the higher elevation had been robbed and the breaking of the overlying strata, of course, naturally allowed the water to reach what we called "subterranean water channels," which in our bituminous coal formation are very numerous.

I recall another instance where we sank a shaft to a depth of 400 ft. and considerable water was experienced and the drill holes penetrated into these fissures. Cement grout was forced into the drill holes up to a pressure of 400 lb. per sq. in. with the result that these fissures were penetrated to a very large area surrounding the shaft. In one instance we traced it to 1800 ft. from the shaft. The day after we finished the grouting a farmer came to us and told us we had spoiled his spring. Upon investigation we found cement grouting in his spring 1800 ft. away.

We find that it is probably impractical or impossible to find a formula which will provide a barrier pillar that is impervious to water. It seems to us that the principal work or endeavor lies in the direction of establishing or suggesting or formulating either a code or a rule which will, in short, safeguard against catastrophe hazards. There is nothing that we can see of greater practical value that can be accomplished.

Dr. Ashley mentioned the strength of a pillar of a few feet in thickness. I know of that instance and of a case where a 6-ft. pillar in the Pittsburgh seam withstood a pressure of 56 ft. of water, and it was perfectly safe. A principal heading had been driven past an encroachment on this property. It was within 6 ft. of this heading. The encroachment was not discovered until after a squeeze was set up in higher workings which crushed the 6-ft. barrier and allowed the water to flood the new workings.

There are other illustrations which I will mention. In large areas of the Lower Kittanning Seam workings, in which pillars are not over 36 ft. wide, these pillars sustain an overburden of from 400 to 500 ft. Recently we drilled a 600-ft. hole, 14 in. in diameter, into the Lower Kittanning Seam 3½ ft. from a rib. There was 590 ft. of water head in the drill hole, yet the water did not break through.

G. S. Rice.—Was there a "muddized" process of drilling, which would prevent leakage of the water through so thin a rib of coal?

C. Enzian.—The bore hole was carefully sand-pumped. It might have become muddized, it is true, but it was a very approximate check as to the water head-resisting strength of the coal. If the coal had been very pliable and had had little lateral strength, the water would have burst out, in fact we expected it would burst out, but it did not.

Another illustration I have in mind is where, at the present time, we have barrier pillars against which variable water heads are present and there is no trouble until we begin to draw the pillars. The moment we break the overlying strats we get an increase in the amount of water.

The singular part of this, which gives us a great deal of hopefulness, is that the water so intercepted will diminish. The fact is that it is actually doing that and the amount of inflow of water is actually diminishing. I do not have detailed figures now but it is quite an appreciable amount. That, I think, is due to the corrosion deposit of sulfur solution which apparently affects these subterranean channels in the same way that water flowing through pipe corrodes it. We feel that the real practical results to accomplish, as Mr. Jessup has said, are to provide against sudden rushes of water and gas and smoke hazards.

In that connection, and in reference to Mr. Jessup's comment on barrier pillars above the water level, my investigations have rather satisfied me that the Commission which deduced the anthracite barrier pillar formula had in mind providing a barrier pillar for mines above drainage level, as well as below it.

As regards testing specimens as an index of pillar strength, a test piece of coal in our big vein in the Georges Creek field (Maryland) would not give you much of an indication of pillar strength because this coal bed has so many slips, or "cutters" as our miners call them. They can never determine in advance where these slips will go.

Speaking of the percolation through the pillars, one of the best illustrations I know of was where the water came through or rather below a barrier pillar, by passing through the fireclay floor and then entering joint planes of a stratum of limestone under the coal, and flowed into an adjoining mine. This, I think, proves that water will percolate through the floor of a seam as well as through its roof.

When it comes to barrier pillars in pitching seams, I seriously question whether it would not be wiser to have shafts in the center of the basin both from the standpoint of safety and from the pumping standpoint. Also in avoiding loss of thick barrier pillars. When barrier pillars are left, I think it is advisable to emphasize that the pillar should be established on each side of the boundary because so many complaints that come to the inspection department are that someone has gone over the line.

R. V. Norris, Wilkes-Barre, Pa.—It may interest you to know in regard to the anthracite ruling that I was present at the meeting of the committee that devised that plan and that I calculated the table which accompanies it. I did not develop the formulas. I was only a cub then. But five times the thickness of the bed was intended for everything above water level. You want further to remember that the beds mined at that time were not 3 and 4 ft. but 7 and 8 ft. thick. We were not mining very much 3 or 4-ft. coal at that date so that the minimum pillar was not 15 ft. but at least 25 ft. While it is true that the calculations went to very much less, the minimum pillar in the minds of the gentlemen who designed it was at least 25 ft. The original law as passed required these pillars to be determined by the mining engineers of the adjoining properties and the mine inspector.

It happened that neither the mining engineers nor the mine inspector had any power to enforce pillar agreements. The mining engineers had no power to tie up their companies—what they said did not commit their companies—and the mine inspectors under the law had no power to enforce them. So the original pillar agreement was a gentleman's agreement made between the largest companies in the anthracite regions. I was present at all of the meetings and I think 1 per cent. of the depth was added to the proposed barrier pillar which was originally planned to be five times the thickness.

G. H. Ashley.—Just one more word in regard to the question of central pumping. We have an illustration of what that may mean in the case of the Soucon Valley mines of Pennsylvania. Many years ago this Institute took a trip to those mines,

particularly to examine what was called the President pumping engine, which at that time was considered the largest engine of the kind in the world. My understanding is that those mines were finally abandoned because they concluded they could not afford to drain all of Eastern Pennsylvania and that seemed to be what they were doing. I am afraid that might be the case in other basins. Many of those basins would be 100 square miles possibly, and if you start at the center and have all that drainage to take care of by a central pumping plant, I am afraid it would be quite impossible financially.

H. Louis, Newcastle-on-Tyne, England (written discussion).—The subject of this paper, namely the necessary thickness of coal pillar to be left to form an efficient barrier against irruptions of water, has attracted considerable attention to this country. In 1927 there was published a report of a Departmental Committee of the Mines Department on the Prevention of Dangers from Accumulations of Water, and a large portion of this report was devoted to the subject of barrier pillars. A very valuable summary of this report is contained in a paper by T. Greenland Davies.[3] I also refer to a very useful paper by Mr. Leeds on boring against old workings likely to contain accumulations of water. Further, I would direct your attention to an interesting paper by H. T. Foster[4] on the disaster due to an inrush of water at Montagu Colliery. In this you will see that a head of something like 200 ft. of water was held back for some time by a rib of coal which in one place was only 6 in. in thickness.[5] I think the only conclusion that one can come to from the information available on these matters is that under some conditions a very thin barrier of coal forms a sufficient protection against a strong head of water, but that under other conditions a very much greater thickness of coal is required and that the only thing that can be done is to keep well on the safe side. The general view in this country appears to

[3] T. G. Davies: Extracts and Recommendations from the Report of the Water Dangers Committee. *Trans.* Inst. Mining Engrs. (1927–1928) **75**, 392. The following in reference to barrier pillars and central pumping stations is quoted from Mr. Davies' paper: Barriers and Dams. (1) The committee agree with the view that the provision of effective barriers is a necessary part of mining engineering practice for reasons of safety. (2) They are of the opinion that the elimination of barriers and the adoption of central pumping stations do not offer a complete cure of the risk from accumulated water. (3) They agree that no definite rule can be laid down to determine the thickness or width of a barrier necessary to retain water, but they do suggest in the report that a barrier which may be subject to pressure of water left in a horizontal seam, or in the direction of the dip in an inclined seam, should not be less (although in many circumstances it should be more) than 20 yards in width.

[4] H. T. Foster: Notes on an Inrush of Water at the Montagu Colliery, Scottswood-on-Tyne, on March 30, 1925. *Trans.* Inst. Mining Engrs. (1927–8) **74**, 41.

[5] Two hewers worked in the holing-bord (room) which was 24 ft. in width on the Friday night preceding the inrush (which occurred on the following Monday), and left a jud (block) of coal 6 ft. wide and 3½ ft. deep, kirved (undercut) on the right side . . .

Fifteen minutes later, the hewer sent for the deputy, who found a trickle of water coming from the face—between the positions of the two shots.—The water burst through shortly afterwards. He . . . turned and escaped.

The holing was afterwards found to be oval shape 7 ft. in width and 1 ft. 8 in. in height. The rib of coal which held the water back for about half an hour was approximately 6 in. in thickness on the right side and 2 ft. 6 in. on the left side of the opening . . .

The level at the point of holing was 208.6 ft. below the level of the water in the Paradise shaft, giving a static head of water of 90.53 lb. per square inch.—Abstract from paper by H. T. Foster, H. M. Inspector of Mines: *Loc. cit.*

be that the minimum thickness of coal left as a barrier should be taken as about 20 yards. No doubt this would be quite safe in the majority of cases, but I am not sure in my own mind whether it is quite safe enough under all ordinary mining conditions.

Discussion at Annual Meeting, February, 1930

H. N. EAVENSON, Pittsburgh, Pa. (written discussion).—If the facts stated in conclusions 1 and 2 of the Commission are correct, and there is no reason to doubt them, it is difficult to see how the proposed thicknesses recommended by the Commission and written in the law were calculated.

In this country, in coal fields where the predominating strata are sandstones and limestones, as is usually the case, it has been my experience that water will go through the strata distances several times the proposed pillar thicknesses with only a few feet of head. In one case in West Virginia, standing water in headings was drained by approaching workings over 300 ft. away and only about 10 ft. lower with no pillar work, and such cases are comparatively common.

In one case in the Pittsburgh Seam in Pennsylvania workings approaching a large body of standing water, under a head of 230 ft., at a distance of 1800 ft. became very wet, the water coming through the coal and also the top, and always drying up about 100 ft. back of the face. This condition persisted until within about 250 ft. of the water where the workings now stand, the quantity of water handled being about constant, and it is believed that if the body were tapped and drained off that no more water would have to be pumped than is now being done. In another case the barrier between an active mine and an abandoned one is 50 ft. There is, and has been for 10 to 12 years, a constant seepage of water through and over the barrier, which shows no signs of failing, but does not keep the water out. The head here is not definitely known. In another case, active workings driving along an old gob line 350 ft. away, with a head not exceeding 70 ft., shows a constant seepage through and over the coal. Other similar cases can be cited.

In general it can be said that under depths of covers up to 700 ft., in the class of strata usually encountered in this country, no practicable thickness of pillar will keep the water out of the lower workings, under even moderate heads.

In such a condition it appears obvious that the pillar is needed only for safety purposes and that a pillar 100 ft. thick is ample for any conditions now being encountered in our bituminous mines, and that one 50 ft. thick is ample for all but a few extreme cases. Such widths will conserve a large amount of our most valuable coals.

Conditions where a lower seam might tap a body of water in workings of an upper seam are rare in this country, as most of such cases are drift mines. In cases of this kind coming under my notice, no trouble has been experienced in working mines in the lower seam, although usually they undoubtedly did drain the mines in the upper one.

H. I. SMITH, Washington, D. C.—May I ask Mr. Ashley whether the water head was taken into consideration in this code, or weight of cover only. I can visualize a great number of cases where you may have a thousand feet of cover and practically no water head, particularly where you are going under a mountain range as in our western country; whereas in dipping beds you may have a much greater water head than there is cover.

G. H. ASHLEY.—As I have indicated, experience seemed to show that a very few feet of coal are sufficient to withstand a very large head of water. In other words, cases were brought up where mining had approached an unknown body of water, and later it had been discovered after the usual advance drilling that there was a pillar only 7 or 8 ft. thick, sometimes sustaining the pressure of 300 or 400 ft. of water, usually, however, at the head of narrow workings. But it became clear early in the

discussion that a pillar of coal of very small thickness would withstand a very considerable water pressure; that it did not act like a gravity dam which sustains the pressure by its own weight, but acts as a beam held between an upper and lower support; and therefore, that the problem was to have enough coal left to maintain the integrity of the barrier pillar under pressure of the overburden, against the tendency to spall and other effects of that kind.

So that the code, as adopted, recognizes first that to meet hydraulic pressure alone, if your coal pillar can be maintained intact, would require a very thin barrier, one only a few feet thick. The larger thickness called for by specifications is to insure that allowing some spalling off of the coal and the breaking down of the roof, and the other factors, that there will still be enough pillar left to maintain itself under those conditions.

Where, as you say, there is no water on the other side of the barrier, it is always possible under the act for the mine inspector and the companies to agree to the removal of the pillar between them. The feeling was that the main thing that must be taken into account is to maintain the integrity of the pillar so that the minimum pillar, after all the spalling, is still strong enough to meet any pressure that might come against it.

G. S. Rice.—The technical data on barriers which have come from Great Britain, and appear in the *Transactions* of the Institution of Mining Engineers (Great Britain), are of great value and rather support the position advanced by Dr. Ashley and others, that a barrier pillar of coal of any considerable thickness will resist a very heavy hydraulic head because acting, apparently, like a beam or rather an arch between the top and the bottom.

G. H. Ashley.—There is one more point in regard to Mr. Eavenson's comment: The Board had the feeling, I think, that beyond the element of safety we should leave to the coal company itself when threatened by water at a higher level, whether or not it should attempt to save itself pumping by leaving a larger pillar. That is, the company would balance the cost of pumping against the value of the coal to be recovered by leaving a thinner pillar and the law should not attempt to specify an amount which could not be estimated, because the conditions differ in every case. As I said, we found so many cases where water was actually following the coal 1500 to 1800 ft., that we felt the question of leaving enough pillar to keep his workings dry must be left to the individual operator. If he thought he could save himself from pumping by leaving a heavier pillar, that was up to him.

H. N. Eavenson.—In the report of the Commission it was stated that in its opinion a barrier 100 ft. thick was sufficient for any conditions, but in the law it is specified that the barrier under the maximum conditions should be 160 ft. thick. Of course, if the adjoining property owners are willing to waive this provision it does not have to be left, but if one of them is not willing the 160-ft. pillar would have to be left along the entire property line. It is generally considered in the Pittsburgh region that a miner on the lower side will have to pump the water sooner or later from the mine on the upper side, practically regardless of the thickness of the pillar, whether it is 100 or 1000 ft. I think general practice now recognizes this fact and the mines go ahead and arrange to pump the water and take the coal out where they have the opportunity.

The new code, of course, has provided that in a case where the operators on both sides are willing to remove the barrier it can be done, but if one of them is not willing to do it a barrier pillar of the maximum limit of width has to be left.

Personally, I cannot see why a pillar 160 ft. in width would resist spalling much better than one 100 ft. wide. I do not believe that either one of them could possibly suffer enough by spalling so that the water would break through, as it is impossible to

conceive that a pillar would spall sufficiently to not leave 20 ft. of solid coal, and, as the figures of the Commission show, this would be plenty to resist breaking through. It seems to me that an extra 60 ft. will be a waste of valuable material, which some day is going to be worth a great deal of money in the Pittsburgh section.

In other respects the law is a big improvement over the old one.

G. S. Rice.—Mr. Eavenson speaks of a 160-ft. width of barrier. That is for a 1000-ft. depth, but suppose you have a depth of 1500 or 2000 ft., as I understand may be possible in Greene County to the lowest coal beds.

G. H. Ashley.—I believe that a clause might very well have been added, making a limit of 100 ft., because I think your criticism is good there. The use of a formula was simply to simplify the statement of the law. The rule and the table, of course, do not appear in the law itself. The table simply shows how it works out, but I believe that a clause stating "up to 100 ft. and not over," or something of that kind, would have been better for these greater depths, and particularly as applied perhaps to other states.

I might say, for the benefit of those from other states who are interested in this problem, that much of the data obtained by the Commission is filed with the Department of Mines at Harrisburg.

In reference to the depth of coal in Western Pennsylvania, the Pittsburgh coal does not go below 1000 ft. in Pennsylvania, but the Upper Freeport is in the neighborhood of 600 ft. lower, and the Lower Kittanning a couple of hundred feet below that.

H. N. Eavenson.—None of those are being mined, are they? Can they be mined?

G. H. Ashley.—They are mining a lower bed at Rock Springs, Pa.

H. N. Eavenson.—At Pittsburgh there is no coal bed lower than the Pittsburgh that is workable. Of course, that may not be true further to the southwest.

G. H. Ashley.—We anticipate that there will be a lot of mining in the Upper Freeport, particularly in Greene County, where it will go well below 1000 ft. in depth, probably 1500 ft. We have little accurate data in that area, however, because there has been very little core drilling. The oil and gas wells commonly report finding coal. They usually report 6 ft. thickness of coal, possibly because that is the thickness drilled between cleanouts. At least there is enough coal found to indicate or suggest that there is a mineable bed at that depth, and many records show coal at still lower depths. So we feel it is quite probable there will be mining at those depths although probably not in our day.

G. S. Rice.—However, whether Pennsylvania has that depth or not to workable coal, we know as deep or deeper coal is going to be mined in other districts in the country; and as this discussion is dealing with the broad problem of barrier pillars, as engineers we are interested in the possibility of its application elsewhere.

A. W. Hesse, Nemacolin, Pa.—One point that has not been brought out should be considered. There is hardly a property that does not have some projecting piece which in order to reach, one must go through a narrow neck; and, if the adjoining mine owner happens to reach and mine out his coal first, you come along and have to provide the barrier pillar called for in this act. It either means that he must stay out and lose it, or that he must, if the adjoining property owner can get to it, sell it at a disadvantage.

I believe there is hardly a coal property owner in Pennsylvania that does not have some little piece, or maybe two or three pieces, of property extend out in that way.

G. H. Ashley.—The code has a number of provisions to meet such cases. For example, it provides that you do not have to leave a large pillar around little inset pieces you may occasionally find in the middle of a property; or a farm that you cannot get, or a farm house reservation, or something of that kind. The code also has this

for such cases: "Provided, that if any of the parties at interest fail to agree on the carrying out of any of the provisions of this Act," and so on, they may appeal to the Secretary of Mines, who shall appoint three reputable and competent mining engineers, who will constitute a Commission, and this commission serves as a board of arbitration.

The thought of the Board was to provide either a direct way of handling these problems, or a way in which these special problems could be met by an appeal to the Department of Mines. I am not quite sure that we have met all the cases, but it was the feeling at the time that the law, as written, seemed to meet those, at least all that the members of the Commission could think of.

A. W. Hesse.—But the possibility is still there.

G. H. Ashley.—I can see how, of course, that under the general wording, you might be stopped from cutting through. But under the provisions of the act if you and your neighbor cannot agree, which means perhaps that you want a certain thing and he does not, you can still appeal to the Department of Mines. Then the Secretary of the Department of Mines will appoint a commission of three engineers and if those engineers agree that you may safely proceed, the law allows you to do that, so there will be no loss.

J. J. Rutledge, Baltimore, Md.—There ought to be some provision made in a general barrier pillar law for barrier pillars starting at the outcrop of the coal seam. I realize that in the area covered by the code under discussion the seams were practically horizontal, but if one has a condition like the original situation in Oklahoma, where the seams pitch from 10° to 60°, and a good part of the openings were at, or just slightly above, the general water level, that is the drainage level, of the region—unless you leave a barrier pillar at the outcrop and continue the boundary barrier pillars down the pitch, the man on the first 40-acre tract worked (and they are all lessees) may work out his 40 acres and go away and leave it, so that the surface water will fill up the mine workings.

The man on the adjoining 40 acres may have more capital. He may work out the coal farther down the dip and yet when the water has filled up this first 40 acres it will come through into his workings, and drown out his mine. In such a case the second lessee has no recourse whatever. This is one of the things that ought to be taken into consideration in problems of this sort. It has happened in two or three cases, and something ought to be said about the continuation of the pillar from the outcrop, especially in pitching seams.

I am very glad the Commission rationalized this pillar idea. Probably one of the reasons for that is that the law requires a reputable geologist on the board to act as counterbalance between the two sides, but the requirement of splitting of the pillar at the boundary line is a mighty good point.

W. H. Glasgow, Harrisburg, Pa.—It might be of interest to those present to know that there have been no appeals made to the Department of Mines as yet. The law has been in effect almost a year, and from experience to date, I feel that the law covers the need pretty well. I hope it will continue that way.

R. D. Hall.—Though this law seems satisfactory for the conditions obtaining in Pennsylvania, the state for which the law was written, it might not serve as well for Nova Scotia or for Harlan County, Kentucky, where bumps occur that actually split the mine pillars open. The late Mr. Herd presented a paper[6] last year in which he shows a rift in a pillar 90 ft. long which was caused by a bump. If any such rift should occur in a pillar which was being used to seal off a considerable head of water it might let a large quantity through in a very short length of time. Consequently

---

[6] See page 15.

it is necessary to concern oneself with the strength of the pillar as a whole. Spalling is not the only action that weakens pillars.

G. S. Rice.—Along the lines indicated by Mr. Hall, I notice that in the discussions which have taken place in the foreign Institutes, a great deal of stress is laid on the advisability of packing waste tightly against a permanent pillar, which they consider will add greatly to the strength of the pillar; and I think that that would be generally the experience of the mining men here, that it would do so. This is probably because back-filling supports and prevents the roof from rupture whereas if the roof shears at the edge of the pillar and the plane of the "shear" or "break" is inclined toward the goaf, then the roof strata will cantilever over the barrier pillar and bring a greater load per unit of area on the pillar than is due to the normal load of the overburden.

Now apparently the strength of the solid coal is very great but whether it would be safe to leave a 20-ft. pillar, for example, without back-filling support, I have considerable doubt. I think the work in the anthracite district has gone to show the importance of loose material as against the edge of a pillar and still more of hydraulic filling adjacent to a pillar.

E. T. Conner, Scranton, Pa.—Many years ago while in charge of an important group of mines in the anthracite region of Pennsylvania, at one of the operations there was some apprehension about the adequacy of a barrier pillar in the principal bed of coal adjoining a mine owned by others where the water was rising. The coal bed in question was about 16 ft. thick. In the adjoining mine the coal remaining in pillars did not exceed 40 per cent. of the original content, while in the mine under my supervision there was about 65 per cent. remaining. The barrier pillar between the two properties was 200 ft. and over.

The adjoining mine had been abandoned because, having extracted too large a percentage of the coal in first mining in the bottom bed some 300 ft. below the Baltimore, a "squeeze" had occurred which opened cracks to what is known as "the buried valley." This is a subsurface valley along the Susquehanna River which had been washed out by erosion long ages ago, and in many instances the "wash" with which this submerged valley is filled is highly saturated quicksand, the depth of which varies from 0 to 350 feet.

Naturally, we watched the rise of water in the adjoining property with interest, to see the effect upon our own operation. The Baltimore Bed varied in depth along the barrier pillar from 250 to 400 ft. below the surface. As the water rose on the other side of the pillar, the first manifestation in our workings was "chipping pillars." This was rather surprising because of the excessive size of pillars on our side of the line. An inspection of the affected area convinced us that no serious damage need be apprehended from the "chipping pillars."

The water continued to rise on the other side until it reached 150 ft. vertical head against the barrier pillar.

About 1000 ft. from the barrier pillar mentioned, there was a tunnel on our side of the line driven through an anticline cutting the bottom rock. Much to our surprise we found a stream of water coming through the strata in the rib and roof of this tunnel, which, as before stated, was below the floor of the Baltimore Bed. This stream continued to increase until it reached a maximum of about 1500 gal. per min. at which it remained practically constant. This caused no particular harm because extra pumps had been provided.

This instance is cited as one of the "freaks" which cannot be foreseen. Ridiculously large pillars may be left in an endeavor to protect adjoining properties or to protect the lives and safety of the employees, but it is difficult, in fact impossible to provide against the "freaks" such as I have mentioned.

Reference was made by Mr. Rice to an experience where pillars "spalled" in old workings and the coal accumulated along the base of the pillar. I have frequently seen such instances, but when the loose coal along the base of the pillar was loaded out, which, of course, is a great temptation to the mine foreman, believing that no harm will result from taking away coal that has "sloughed" off the pillar, chipping again began and seriously impaired the strength of the pillars. In other words, it is usually not safe to load out this relatively small value in the matter of support, because such action may be a case of "the last straw breaking the camel's back"—taking away this relatively small support from pillars may cause the overburden to crush the pillars entirely.

The commission associated with the author in the drafting of his paper embraces men of large experience, and undoubtedly they have looked up all of the available data. As far as I can see, I believe they have covered the ground as thoroughly as they possibly could.

I have seen other instances of what I have termed "freaks" in the opposite direction, *i. e.*, of pillars of coal in both anthracite and bituminous mines that appeared to be ridiculously small and yet which withstood tremendous water head, so that I am of the opinion that we should be careful, as suggested by Mr. Eavenson in not going too far in prescribing "hard and fast rules" that may mean large losses of available coal in an endeavor to protect ourselves and to protect life. The subject should be thoroughly considered both ways, because, as I have before stated, there are "freaks" in both directions that must be taken into consideration before enacting into laws that cannot be changed, rules which may inflict serious hardship without compensating benefits.

H. N. Eavenson.—I would like to call attention to the statement in the report of the Commission of the fact that they have never known a case where a pillar, even as low as 10 ft. thick, had failed on account of the water pressure. When you think of the multitude of seams that experience covers, and the multitude of conditions with high heads of water, and with the way coal is deposited, and what we know about its character, it certainly must add to our respect for this material that an occurrence of that kind has never happened.

G. S. Rice.—We are conducting some very interesting tests in the Bruceton Experimental Mine on the strength of coal in place to resist the thrust of solid concrete stoppings acting as an arch, using a hydraulic jack screw. Preliminary results indicate that the coal when pressed against the "faces" does not crush until a pressure of 14,000 lb. per sq. in. is reached. We had found prior to this, when massive concrete stoppings, acting as an arch, failed that in no case did the coal of the rib buttresses show any signs of crushing; it was always the concrete which gave way by crushing in the middle of the span; yet that same coal, if you take it out in a lump and put it into a testing machine, will crush 2000 to 3000 lb. a square inch.

H. N. Eavenson.—I would suggest to Dr. Ashley that if he has an opportunity, he have someone compile the figures that he has in his possession showing the thicknesses of pillars, the amount of head against them, and all pertinent data, so that these data will be available to the profession at large, and they could be submitted in the form of a brief paper. We understand that these are all on file in his office, but if the salient facts could be condensed and published it would be a very great advantage to the profession.

Subsidence from Anthracite Mining

By H. W. Montz,* Wilkes-Barre, Pa.

With an Introduction on

Surface Support

By R. V. Norris,† Wilkes-Barre, Pa.

(New York Meeting, February, 1928)

SURFACE SUPPORT

By R. V. Norris

The problem of surface support in coal mining is naturally divided into three branches:

1. Surface covered with improvements of such value as compared with the value of the underlying coal, or with such elements of danger to mining, that absolute support is essential.

2. Surface with improvements of less value than the underlying coal such that moderate settlement may be permitted, but destructive settlement must be avoided, either by reason of danger to human life or unwarranted property damage.

3. Surface of relatively small value as compared with the underlying coal where destructive settlement may be permitted.

First Problem

In the first case it is evident that the original support must be maintained either (*a*) by leaving ample pillars and confining the coal recovery to a very conservative first mining, (*b*) by so supporting less conservative first mining by filling or flushing that the gradual deterioration of the pillars may be prevented and subsidence avoided, or (*c*) by incompressible support as masonry either as an adjunct to insufficient pillars or in place of the original coal support.

a. From tests made by the Scranton Engineers Club, it appears safe to figure on 2000 lb. per sq. in. as the safe squeezing strength of coal pillars in place, provided that these are, as usual, not higher than their least horizontal dimension; and from this figure the percentage of pillar to be left for various depths may be easily calculated. For safety in this

* Mining Engineer, Lehigh Valley Coal Co.

† Consulting Engineer.

case it would be necessary to protect the pillars from deterioration by lagging, or better, by packing, and for minor depths the roof between the pillars should be at least partly supported by packing or flushing to prevent local caves.

b. With full support as by flushing or careful packing it seems probable that as high as 3000 lb. per sq. in. would be safe for coal pillars in place, and that the first class support could be given by a percentage of pillars calculated on this basis with full flushing.

c. As the materials available for masonry in the mines are usually relatively weak sandstones and slates even first class masonry supports built of these will have no greater strength than the original coal pillars, and as the masonry would cost at present prices much more than the value of the coal in place, it does not seem practicable to remove the coal and replace by masonry supports where the first class surface support is required.

Second Problem

The second case, where moderate settlement may be permitted, but destruction of the surface should be avoided, is the general condition applying to ordinary built-up territory without monumental buildings or special mining conditions. In this it is assumed that all possible coal is to be removed and the problem becomes one of partial support and the avoidance as far as practicable of destructive fractures. This, then, requires such mining and support as will permit gradual settlement, and while it cannot be absolutely accomplished it can be approximated by two methods:

a. By leaving sufficient pillars for temporary support, which will gradually disintegrate under pressure, avoiding the leaving of any large blocks of coal which by their resistance would cause a breaking off of the strata and fracture of the surface. Such mining would necessitate very evenly distributed pillars of practically equal size, and the expectation of a general squeeze and final loss of the pillar coal thus sacrificed. That such squeezes may result in surface settlement without serious injury is shown by a study of the results of many squeezes in the mines of Lackawanna and Luzerne counties, the surface damage from which, assuming sufficient depth to prevent cave holes, has usually been confined to the vicinity of large blocks or of barrier pillars.

One of the very serious mistakes made by the public is in insisting upon the leaving of large permanent pillars to protect certain buildings, roads, and so forth. In repeated instances within my own practice the leaving of those pillars has resulted in destructive surface injury where a general settlement did not do so.

There is one particularly interesting case of a very large general settlement, to which I think the author will refer, where a general squeeze

Wm R. Chedsey.

carried down the surface at one rush, something like 3 ft. breaking off at the edges of the unmined coal. At that cut-off line there was a distinct fracture and the workings were in some places very deep. That whole property, 800 to 1000 ft. or more wide, went down so quietly that even brick buildings on it were not appreciably injured. On the other hand, I know of repeated instances where the leaving of a single large pillar for the support of some special property resulted in the absolute destruction of that property from side pull.

b. By extensive flushing or rock filling, the coal should be capable of full removal; the inevitable settlement on to the filling which, as shown by Conner and Griffith, may compress from 25 to 30 per cent., should be gradual with a minimized surface damage if second mining is carried on regularly and systematically. A careful study should be made of the results to the surface of the extensive second mining now in progress, with the purpose of determining the amount of filling necessary to prevent destructive rupture of the strata. I believe it will be found that very much less than full filling by flushing or packing is necessary to attain gradual settlement, with a minimum of surface damage, provided that the second mining as it progresses is complete, and that no solid blocks are left to form breaking points for the strata.

Third Problem

The third case, where destructive settlement is permissible, covers the greater part of the minable area. It would seem a mistake to use the limited amount of flushing material in territory of this sort, and the filling desirable for mining purposes only should, in such territory, come from the mine rock rather than from finer material.

Surface Beds

This discussion presupposes a sufficient depth of workings to prevent breakage of the surface by local roof falls. Unfortunately, in many places large beds lie so close to the surface that simple roof falls in the workings result in disastrous surface rupture.

Where the surface value warrants the expense, such shallow beds should certainly be supported by full flushing or filling regardless of the conditions in the underlying beds, and it is my opinion that such support would go very far towards minimizing or even preventing surface damage from subsidence in the lower beds, as the fully flushed workings in surface beds, even with a full removal of coal and reflushing of the openings with the resulting gradual subsidence, should act as a cushion to distribute the settlement from the removal of the coal in lower beds, and minimize the surface damage therefrom.

Summary

In my opinion, complete support can be attained only by leaving sufficient coal pillars properly protected, or the replacement of such pillars by first class masonry of sufficient strength to carry the overlying strata. But damage may be minimized and in the main avoided by the support by flushing or filling of any surface beds that may threaten surface damage from local falls or cave holes, and by bringing about gradual subsidence of the underlying beds by complete removal of the coal with partial support from packing or flushing, with the removal in second mining of all blocks or pillars which could by their resistance cause fracture of the strata. Further, sections of second mining should be so chosen that the edges of the area, where fractures and destructive surface damage are inevitable, will fall in places where such damage is permissible.

Separation of Coal Title from Surface Title

There is another interesting point in connection with the subsidence in the anthracite region; that is, the very general separation of the coal title from the surface title. The coal in most instances was sold from under the surface and that sale, being made when the surface had relatively small value, was usually an absolute sale of the coal with a release by the surface of all damage from mining of any sort, shape or description. The courts have upheld that position where it was properly taken in deeds, but the public has not, and it has become the practice of the mining companies, as a matter of public policy, to repair the damages on smaller dwellings and smaller properties at their own cost. They are not so willing to repair damages on larger buildings of which the owners are able to protect themselves, and where the value of the coal is not many times the value of the building.

SUBSIDENCE FROM ANTHRACITE MINING

By H. W. Montz

A further study of the effect upon the surface of mining operations in the Anthracite Region of Pennsylvania has become imperative as an economic measure for future conduct of the industry. The subsidence is not only of local interest; it has attracted much and widespread attention, even beyond the confines of the region. The growing communities, towns and cities are vitally concerned; also allied interests such as railroads, manufacturers and so forth.

In 1911, two eminent engineers, Eli T. Conner and the late William Griffiths, were retained by the City of Scranton to report upon the mining conditions under that city. Their report contemplated a study

of mining conditions under the city with a view of minimizing subsidence by means of artificial support such as flushing.

In the same year, Governor Tenner appointed a commission to study the mining conditions in the region with respect to surface support, and that report, made in 1913, resulted in the enactment of the Davis mine-cave law.

Although other reports have been made in connection with subsidence, among which the most complete, no doubt, is that of L. E. Young and H. H. Stoek, published as University of Illinois *Bull.* 91, the first two are the only ones, to my knowledge, that treat solely anthracite problems. Their investigations, however, were an attempt to preserve the support.

Purpose of Study

A very considerable portion of the surface in the anthracite region has been more or less disturbed by robbing operations thus far; although the policies of the various companies have been to confine these mining operations, so far as possible, to outlying and unimproved surface areas. The real problem, however, has yet to be solved, so that operations may continue with attendant prosperity to the industry. Whether the right to disturb the surface is real or otherwise, the operators feel a moral responsibility, if not a legal one, to minimize surface damage; therefore an economic problem presents itself for solution, in which the study of the subject of this paper should be an important factor. In the consideration of such problems, it would certainly be not only interesting but pertinent to predict the probability and extent of surface damage.

Realizing that a study of this kind requires years of investigation, particularly in acquiring data that are available only through observed experience, this attempt is only a preliminary study, with the hope that sufficient observations of surface movements and effects, together with contributing mining conditions, may be and will be made in the future, so as to permit, to some degree, the determination of expectancy. Thus far, this has been impossible, because of the lack of pertinent data, and while it is doubtful whether exact formulas can be deduced to calculate definite surface disturbance from certain mining conditions, it is probable that the results can at least be classified.

Method of Mining in Anthracite Region

The general practice of mining in the anthracite region has been the room or chamber and pillar method. While there are other systems in operation, such as longwall and other modifications, the first named predominates.

The gangways are driven 12 to 14 ft. wide, with chambers at right angles, or nearly so, driven 20 to 24 ft. wide. These chambers or "breasts" are driven on various centers, ranging from 40 to 70 ft., thereby leaving a pillar also varying in width from 20 to 50 ft. It is from the second mining or robbing of these pillars, generally speaking, that subsidence results. Another source of movement is from "squeezes," which often occur incident to too great an extraction on first mining and which often attend on robbing operations. The "squeeze" often results in a complete collapse of surrounding territory. All of these attending results depend largely on the physical conditions, such as the nature of the overlying roof and strata, the thickness and character of the vein, pitch, etc. As mining is carried on and as roof conditions require, the openings are timbered or propped to support the immediate roof particularly for safety.

When the pillars are sufficiently wide, the second mining is carried on by driving narrow holes, approximately 10 ft. wide, through the line of pillars, until the point of retreat is reached. The pillar is often skipped or sliced, as in the cases cited, when conditions are favorable and permit. The actual robbing is then started by the removal of the remaining part of the original pillar. As the retreat is made, the roof rock starts to break for lack of support and thus the operation is carried to conclusion. In order to hasten the caving of the roof, the timbers are often drawn and salvaged. The roof is often blown, for safety as well as to assist in the relief of pressure on the surrounding pillars.

Examples of Subsidence

As there are numerous examples of subsidence from mining operations, a great many could be cited, but, generally speaking, the pertinent data such as would be necessary in the consideration of this study are not available. Two areas that have afforded, to some degree, an opportunity for observation, each under different physical conditions, at least as far as the surface is concerned, are shown below as Exhibits I and II. Exhibit III belongs to the class to which I have just referred; namely, in which available information is interesting only as a common example of subsidence.

Exhibit I is an area under which robbing was contemplated in regular sequence and also under improved surface. It is adjacent to areas under which extensive robbing had been carried on with but slight subsidence to the improved surface, visible only here and there to a very limited extent. Furthermore, it had been well flushed or filled with silt in the original openings from first mining in the surface vein.

Exhibit II is an area in which no flushing was done previous to the robbing of the pillars and over which the surface was not improved,

although adjacent to improvements. The robbing, as in the first case, was carried on in the surface, or top, vein. The log of overlying strata (Fig. 4) shows considerable wash overlying the rock, with a large proportion of sand in each case. It is extremely difficult to form conclusions with any degree of certainty, on account of the great dissimilarity of the wash as compared with the bed rock.

Exhibit III shows the subsidence over an extended period for more than 20 years, in which time the underlying veins had not only been first-mined and robbed to a large extent, but had been subjected to accompanying squeezes as well.

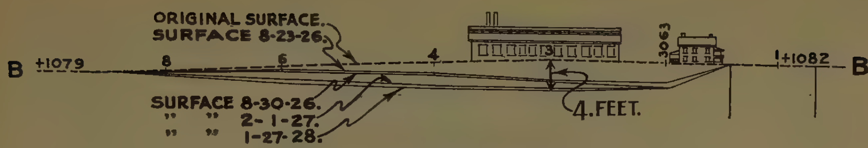
Monuments

In the case of Exhibit I, iron pins were "planted" on the surface, approximately 200 ft. apart and in squares, to ascertain the subsidence vertically by taking observations at regular intervals and to discover the lateral movement. Unfortunately, these observations were not made at intervals short enough to determine with necessary accuracy just when and how much of the movement occurred during short stated periods of time.

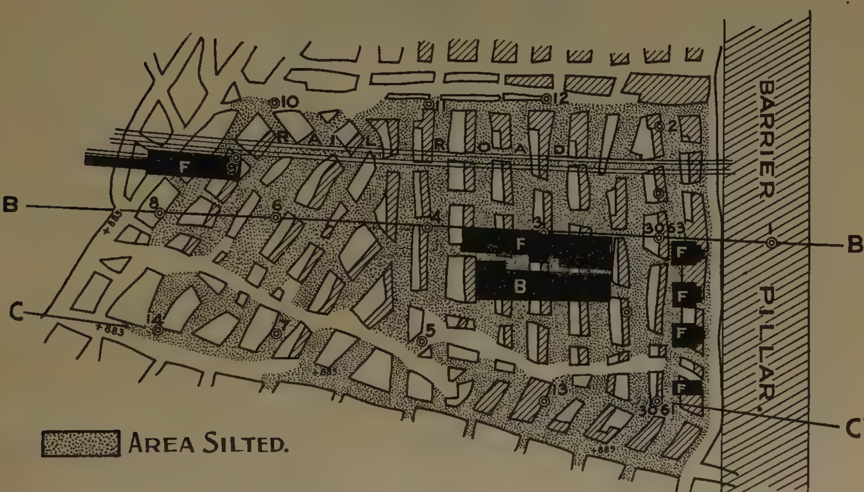
At present, there are other areas where robbing of pillars is being contemplated and over which monuments have been placed for observation.

Description of Areas

The area referred to as Exhibit I (Fig. 1) covered approximately 7 acres overlying an area of surface vein which was mined over and in which the pillars were about to be robbed. The vein, practically flat, was approximately 7 ft. thick as shown on the cross-section in Fig. 2, and was well filled with silt. The surface, correspondingly flat, was improved with streets, a railroad and several buildings, all of which are shown on Fig. 1. The easterly boundary of the area lay adjacent to a substantial barrier pillar. A typical columnar section (Fig. 2) beneath this area showed 15 ft. of gravel, grading into sand and gravel, which continued for another 15 ft. Below this is 40 ft. of brown sand, becoming finer with depth, to a typical quicksand, and then a streak of hard clay, 27 ft. thick. Underlying this occurred slate, thin coal and bone in alternate layers, then approximately 9 ft. of fine conglomerate, grading to a sandstone 76 ft. thick. Directly underneath this rock is the surface vein. Overlying the vein and adjacent to it is a slab of sandstone, averaging probably 8 ft. in thickness, which is parted from the main sandstone top and which falls as soon as the pillar is removed. The slicing or skipping of pillars had been completed and the retreating started when on the evening of Aug. 21, 1926, a maximum subsidence of 2½ ft. took place. On the morning of that day, the area was apparently "on the move,"



CROSS SECTION.



PLAN OF MINING.

OBSERVATIONS OVER SUBSIDED AREA.									
STATION AND I.P. №	ORIGINAL ELEVATION 3-29-26.	8-23-26	SUBSIDENCE. FT.	8-30-26	SUBSIDENCE. FT.	12-1-27	SUBSIDENCE FT.	1-27-28	SUBSIDENCE. FT.
3061	1079.812	1079.735	- .077	1079.735	- .077	1078.091	-1.721	1078.044	-1.768
3062	1081.010	1081.010		1081.010		1081.010		1081.010	
3063	1082.197	1079.648	-2.549	1079.637	-2.560	1079.071	-3.126	1078.975	-3.222
3577	1083.094	1080.798	-2.296	1080.744	-2.350				
I.P. 1	1081.818	1081.809	- .009	1081.838	+ .020	1081.807	-.011	1081.828	+ .010
2	1084.690	1083.084	-1.606	1083.088	-1.602				
3	1082.539	1079.828	-2.711	1079.765	-2.774	1078.750	-3.789	1078.535	-4.004
4	1081.788	1080.374	-1.414	1080.295	-1.493	1078.622	-3.166	1078.381	-3.407
5	1073.026								
6	1081.031	1080.763	- .268	1080.756	-.275	1079.377	-1.654	1078.950	-2.081
7	1078.211	1077.790	-.421	1077.787	-.424	1076.969	-1.242	1076.739	-1.472
8	1080.313	1080.101	-.212	1080.063	-.250	1079.829	-.484	1079.662	- .651
9	1082.399	1082.264	-.135	1082.259	-.140	1081.335	-1.064	1080.891	-1.508
10	1080.322	1080.146	-.176	1080.137	-.185	1078.679	-1.643	1078.370	-1.952
11	1090.483	1089.460	-1.023	1089.397	-1.086				
12	1086.755								
13	1073.759	1073.307	-.422	1073.294	-.435	1071.657	-2.072	1071.445	-2.284
14	1075.404								

SCALE IN FEET.
0 50' 100'

FIG. 1.—EXHIBIT I. CHECKER VEIN ROBBING.

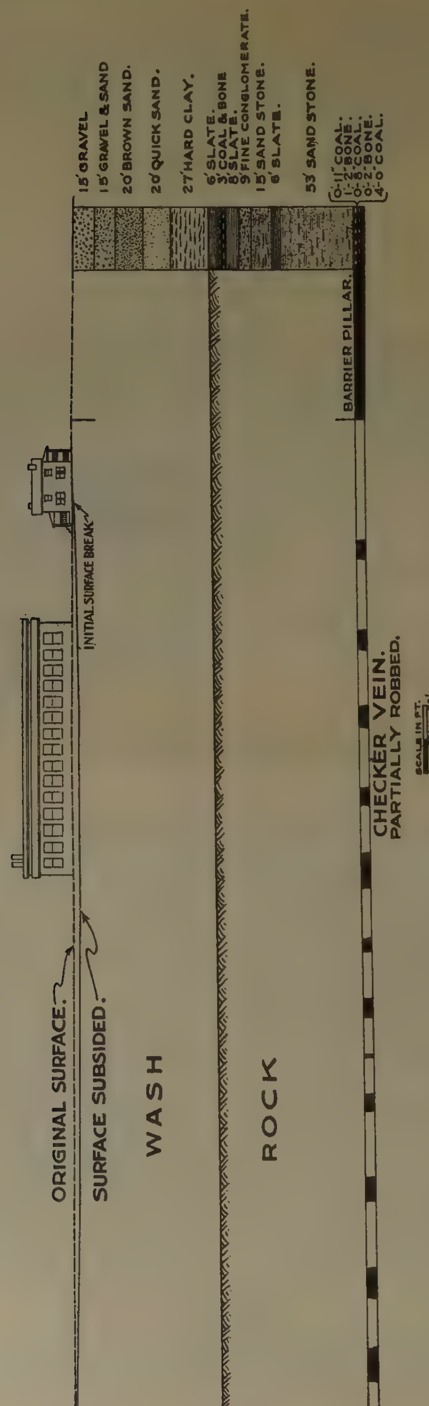


Fig. 2.—Cross-section through BB' of Fig. 1.

there being a heavy squeeze, and no work was allowed in the territory that day. The squeeze continued, with the resultant collapse in the evening.

The houses along the easterly side of the street, adjacent to the barrier pillar, were pitched forward and badly shaken. The frame and brick structures had settled rather evenly and only small cracks at the ends of the buildings were noticeable. The railroad tracks were likewise settled but brought to grade immediately. Since that time there has been a slight but continuous settlement, so gradual as to cause no additional damage to the buildings. I have no doubt that the passing of heavy trains over the area aggravated the squeeze and hastened the collapse.

Another interesting feature in connection with the subsidence is the fact that monument No. 1, which was placed over the barrier pillar, is slightly higher in elevation now than it was when originally installed.

Exhibit II (Fig. 3) is likewise an interesting case of subsidence. In the vicinity of the robbing, there is a paved avenue with a number of houses facing the street. The ground is generally level and continues practically level to the rear of the buildings, where there is the top edge of a bank 40 ft. high. The surface from the foot of the bank is practically level. A typical columnar section beneath this area (Fig. 4) shows a bed of sandy gravel, averaging 60 to 70 ft. thick, then a streak of clay from 15 to 20 ft. and 50 to 60 ft. of fine sand overlying the bed rock. The rock underlying the wash is a hard sandstone and extends 160 ft. to the Four Foot vein, which is 6½ ft. thick and practically flat in pitch. The robbing, as shown on Fig. 3, was a continuation of robbing operations from the west, from which there had been no visible signs of disturbance on the surface. The robbing in this particular area started in June, 1925, and it was not until Oct. 29, 1926, that there was any evidence of disturbance. On that date, a sudden subsidence of a few inches took place and affected two of the buildings to the extent that the doors and windows tightened. In three days the foundations of at least four of the houses were cracked and from that time on the subsidence was a gradual process, with cracks appearing on the surface. After the first few days, however, the movement, instead of being a direct subsidence, seemed to be a pull to the south. Undoubtedly the movement was aggravated largely by the 40-ft. drop of the surface at the rear of the improvements. As the cracks in the surface appeared, the subsidence of the buildings was very much more pronounced to the south; on two of the buildings, which were of brick construction, the rear dropped off very abruptly, so that the houses were approximately 1 ft. lower in the rear than in the front. The greater amount of damage, however, seemed to have been confined to cellar walls. On March 11, 1927, a small crack appeared in the brick pavement in the avenue, directly over the barrier pillar, and since that time there has been a

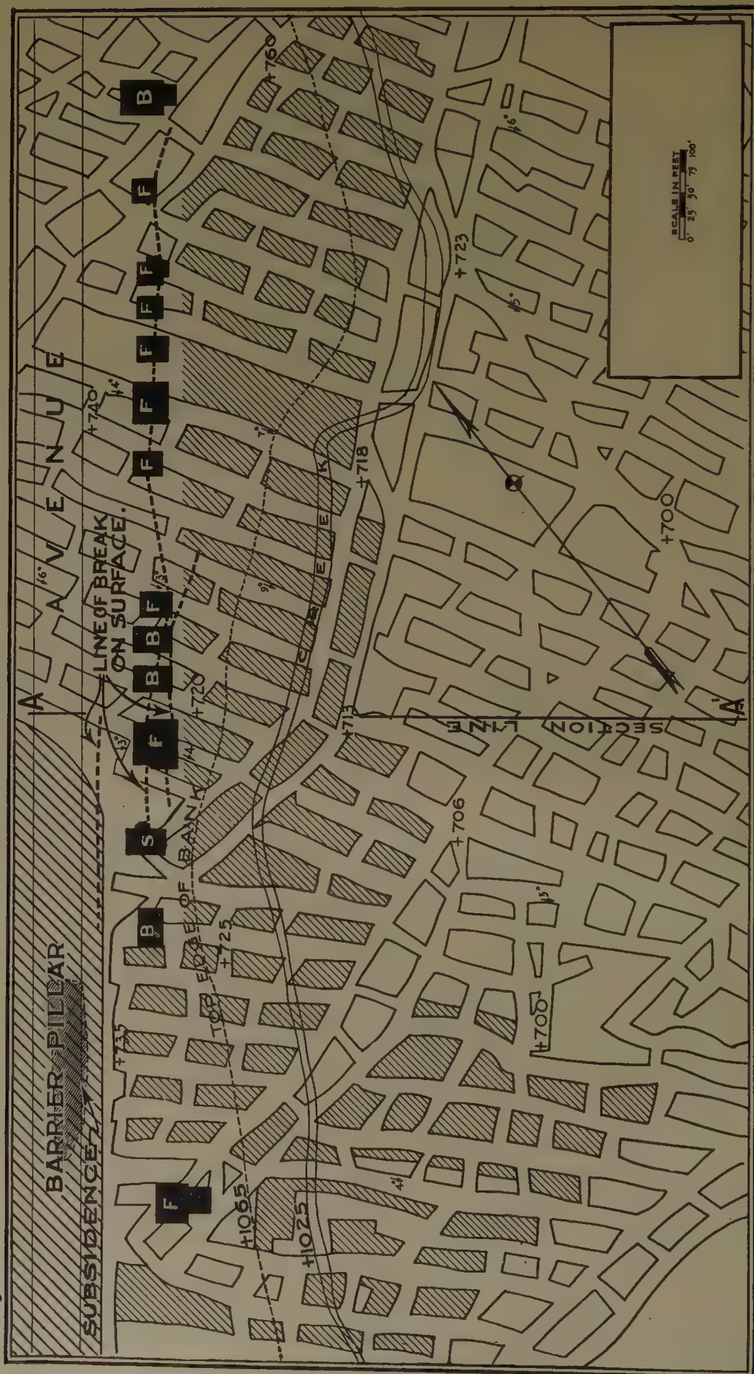


FIG. 3.—EXHIBIT II. FOUR FOOT ROBBING.

continual slight settlement over the area shown cross-lined (Fig. 3) to a maximum settlement of 6 in. This settlement under the avenue is taking place notwithstanding that the coal under the area is solid. Subsequent to Oct. 29, 1926, the date of the first subsidence, there was a constant movement over the affected area until June, 1927, when nearly all movement ceased. It remained in that state, or apparently so, until October, 1927, when further movement was detected in the area that had settled most severely the year previous. This continued for several weeks, then apparently stopped, and no movement has been detected since that time. As soon as the subsidence was discovered, the robbing in that territory was stopped and the pillars that were reserved for support under the houses and street are still intact and still in normal condition.

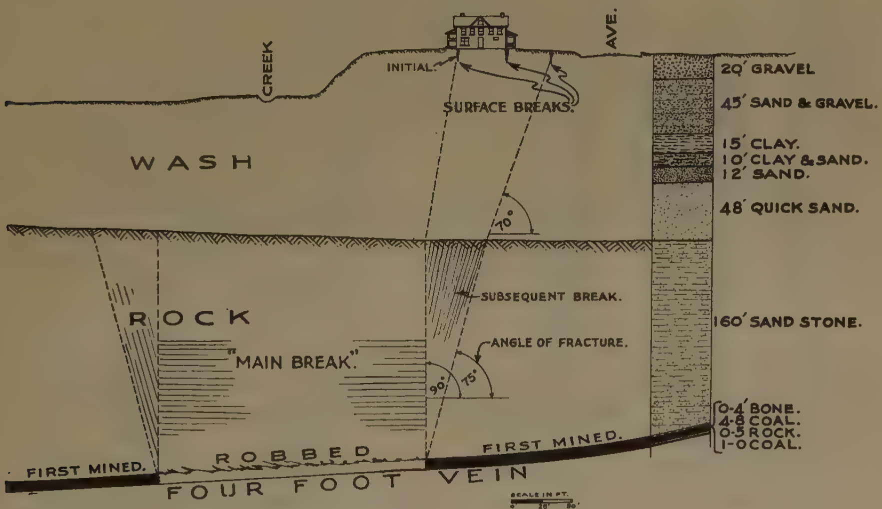


FIG. 4.—CROSS-SECTION THROUGH AA′ OF FIG. 3.

Exhibit III (Fig. 5) is interesting only from the point of view of what the ultimate subsidence would be under similar conditions. The only data available are in the record of surface elevations taken in August, 1906, which at that time had been subjected to subsidence as a result of a squeeze in the lower veins. The total depth from the surface to the bottom vein was approximately 800 ft. and the veins that had been worked at that time were the B, C, D, E and F, the F vein being approximately 535 ft. below the surface. The territory had been first-mined and approximately 50 per cent. removed at the time of the squeeze, which completely crushed the coal and made it an unprofitable proposition for further mining. Later on, however, the upper veins were opened, first-mined and robbed to the point where large areas have now been permanently abandoned. The surface has been subsided a maxi-

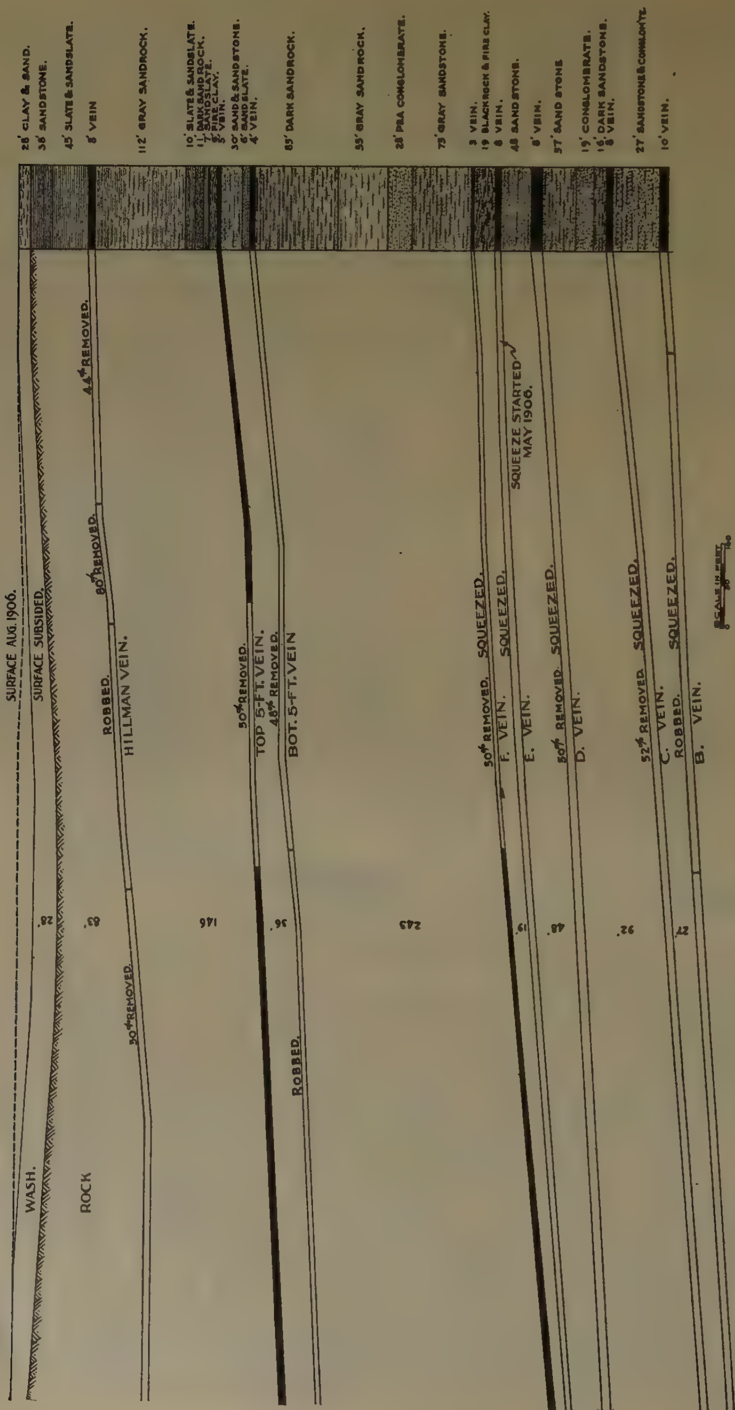


FIG. 5.—EXHIBIT III. CROSS-SECTION THROUGH ROBBED AND SQUEEZED AREA.

mum of 20 ft. and, although not visible, subsidence is still taking place. Very little damage has been sustained by the improvements located on the surface, which consisted of dwellings, a railroad and a number of smaller buildings.

Deductions from Observations

From Figs. 6 and 7 it appears that after the initial movement occurred, it continued at a uniform rate for a period of 1½ years without having reached a state of rest. The initial settlement varied from 3 in. to 2 ft.

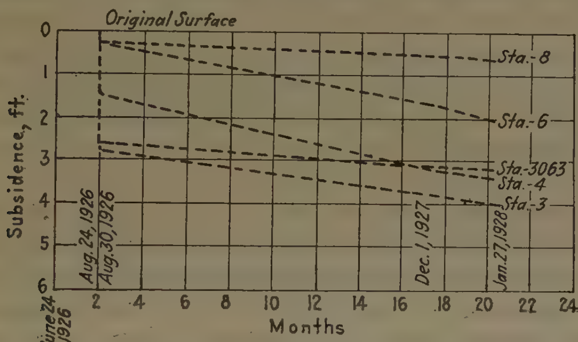


FIG. 6.—TIME SUBSIDENCE CHART, LINE BB′ OF FIG. 1.

8 in. and since that time the observations indicate a subsidence rate of 1 in. per month. Of the total to date, the initial, as compared with the total subsidence, varies from 12 per cent. minimum to 80 per cent.

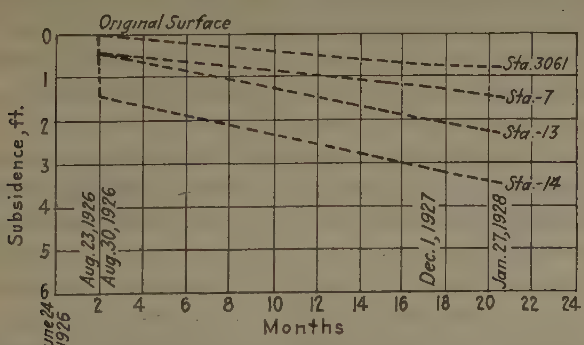


FIG. 7.—TIME SUBSIDENCE CHART, LINE CC′ OF FIG. 1.

maximum, the average of all observations being 41 per cent. It is to be noted, however, that where the maximum occurred at the initial break, its rate of continued subsidence was substantially decreased. A further deduction from the observations taken along line *BB′* (Fig. 6) indicate

the greatest movement in the middle of the affected area, with a decreasing subsidence radiating from the center.

The conjectures of engineers as to what actually occurs in relation to the break in the overlying strata have been at variance. Hausse, a German engineer, made a scientific investigation of subsidence in which he classifies the breaks as the "main break" and the "after break." He says that over horizontal beds the main break is vertical and the after break extends over the pillars. He further assumes the angle of after break to be constant or equal to 20°, decreasing from 20° to 10° in proportion to the increase of the dip from 0 to 45°. In a report made by the Board of Mines of Dortmund, they found that the lateral extent of subsidence greatly increases with thickness of wash or "marl" covering. From the data obtained, they determined the angle of fracture from bed rock to surface to be approximately 70°, measured from horizontal; while in rock dipping not more than 15°, the angle of break was found to be 75°.

On the cross-section through AA' (Fig. 4), shown in Exhibit II, the German theory is illustrated, indicating the main break, the after or subsequent break, and the angles of fracture through bed rock and marl, or wash. Whether or not the surface break in this instance was coincident with the theory, the writer is unable to say. It is his opinion, however, that in the anthracite field, the angle of fracture more nearly approaches the vertical through stratified measures of rock horizontal or nearly so, which is borne out by numerous examples. It seems reasonable to suppose that the overlying wash, stratified as it is with gravel, clay, sand and particularly quicksand, will assume an angle of fracture considerably less than 70°, which in this case would be 50°, on the theory of the vertical break through the bed rock. The Law of the Normal is a generally accepted theory; it has limitations as to its application to the heavy pitches but under the lighter pitches it seems to apply, as in this case.

A further deduction from the results of the observations made in connection with Exhibit I is the fact that the total subsidence has reached a maximum of 4 ft., which is 57 per cent. of the thickness of the vein. It must be appreciated, however, that at the time that this subsidence occurred 86 per cent. of the original coal content had been removed but 46 per cent. of the total area had been well silted. In other words, the natural support of the remaining pillars plus the artificial support, amounted to 62 per cent. of the whole. A perfectly natural conclusion is the fact that the artificial support of silt was an auxiliary support which tended toward a reduction in subsidence only. It is to be admitted, however, that had the actual removal of pillar support been confined to a more limited area, namely, a more limited robbing face, the subsidence would not have been aggravated to the point experienced.

Conclusion

Recently, some additional data have become available in connection with subsidence on heavy pitches, but the behavior of disturbance to overlying rock strata has been so varied, which has been the experience in other mining regions also, that further observations should be made before this subject is elaborated.

It is quite apparent that the effect of subsidence of underlying veins upon overlying veins has not been considered, although it certainly comes within the scope of subsidence intended to be treated in this paper. The data available for such consideration are practically nil and this phase must therefore be taken up in the future. In a general way, the experience in this regard has been decidedly varied, depending largely on the physical conditions, such as the thickness of the underlying veins, thickness of rock interval, character, depth below surface, etc.

There are examples where the underlying vein has been robbed leaving the overlying vein virgin and the latter when subsequently developed has been found not to have sustained a permanent injury. The overlying vein was certainly disturbed and subsided, but ultimately the vein was found placed relatively in its original formation and position. Cases have also been found in which the overlying vein was subsided in comparatively small areas in the virgin but the overlying roof did not yield, the parting being as much as 6 in. between the roof and the top of the vein.

Where the underlying veins have been robbed under areas of overlying veins, which have been first-mined, the experience has been varied but, generally speaking, the overlying vein has been rendered unminable.

Knowledge as to this phase, however, is secondary to the study treated in this paper and therefore is only incidentally mentioned.

An effort must be made to acquire data along these lines as promptly as possible, and as the result of a carefully laid out program, so as to obtain all the data incidental to the study. It is suggested to the Institute that a committee of anthracite mining engineers be appointed to cooperate with the Committee on Subsidence in the further study and discussion of this subject. The information should be acquired along the following outline:

1. To select for mining out areas that offer a reasonable probability of obtaining informative data.

2. Such areas should be large enough to prevent local disturbances from being given undue weight.

3. Some areas should include the outcrop where it is in the immediate vicinity, to indicate areas of most destructive surface disturbance.

4. The areas should be not less than 500 ft. wide and under normal conditions not less than 6 acres in extent, sufficiently large to overlap on all sides a proposed second mining area.

5. Monuments should be placed on the surface on 200-ft. squares and these should be located both horizontally and vertically from permanent bases outside of any probable area of disturbance.

6. Original surveys and levels locating these monuments should be made with careful cross-sections in both directions, with proper record maps filed. These surveys and levels should be run at regular intervals, at least once every two months.

7. Original surveys of the mine workings underlying and in the vicinity of the observations should be filed.

8. To coordinate the mining conditions with the surface observations, the mine workings and test areas should be resurveyed simultaneously.

9. Full notes should be kept of both mining and surface changes, so that proper deductions can be made.

10. If surface improvements are involved, in addition to the above, full data as to the condition of these should be obtained at the start and complete records should be kept of settlement or damage.

11. The effect of any disturbances in the mines on any intermediate underlying or overlying beds should be noted.

Acknowledgments

In developing this study, the author has had the cooperation of the following executives: J. M. Humphrey, president of The Lehigh Valley Coal Co.; C. F. Huber, president of The Lehigh & Wilkes-Barre Coal Co.; W. W. Inglis, president of the Glen Alden Coal Co.; J. B. Warriner, general manager of the Lehigh Coal & Navigation Co.; R. H. Buchanan, president of the South Penn Collieries Co.; E. H. Suender, general manager of Madeira, Hill & Co.; T. E. Snyder, vice-president and general manager of the Hazle Brook Coal Co. R. V. Norris, consulting engineer, also gave his very able counsel and assistance.

DISCUSSION

R. V. Norris, Wilkes-Barre, Pa.—Mr. Montz, chief mining engineer of the Lehigh Valley Coal Co., at the request of the Anthracite Section attempted to collect and codify some of the data available on subsidence in this region. It might seem very easy, where there is constant subsidence everywhere, to get these data. One can see cave holes and cave damage everywhere but it is almost impossible to find the cause and extent of the damage. Except where special arrangements have been made to watch it, in most instances there is no reliable base to measure from and there are no reliable measurements from a reliable base of the original surface conditions; so while it is known that there has been subsidence, the amount is not known, and unless it is watched very closely its relation to underground difficulties is unknown.

G. S. Rice, Washington, D. C.—Do you make any technical distinction between robbing and removal?

H. W. Montz.—We call a robbing operation any operation which actually removes the entire pillar. That necessarily has to be done in two operations: (1) the skipping of the pillar or driving the pillar hole, and (2) the actual robbing, the final removal of the remaining pillar.

R. V. Norris.—In practically every case the destructive damage to buildings on the surface has been along the line of either barrier pillars or unrobbed workings. The destructive damage has not been over the main body of the robbed area except on its edge.

E. T. Conner, Scranton, Pa.—This subject is so vast that the work that was done by my colleague, Mr. Griffiths, and myself can only be considered as rather fragmentary. We found, when we were confronted with the job of describing the mining conditions beneath the city of Scranton, which might and did affect surface improvements, that the information was extremely fragmentary and indefinite. In fact, the earlier plans for mining in that territory usually contemplated extracting about two-thirds of the coal and leaving the balance in pillars rather irregularly disposed with no expectation of recovering any portion of the pillar coal. We satisfied ourselves that that rule was reasonably safe where the beds lay no more than 200 ft. below the surface. The situation varied, however, with the character of the coal bed, and in beds of greater depth the rule of leaving one-third for the support of the overburden did not apply.

Mr. Griffiths and I were well satisfied with the statement made by Mr. Norris and concurred in by the author of the paper, that the greatest disturbance of the surface occurred along the edges of solid blocks of coal, barrier pillars or so-called protective pillars. In many instances the original owner of the coal, in the erroneous belief that he was protecting himself, specified that the coal beneath his surface improvements should not be mined. The result was that blocks that had been left to support surface improvements were scattered all over the territory that we at that time examined. In almost every instance those blocks of coal resulted in damage to the person who thought he was protecting himself.

Mr. Montz has presented some valuable data that should be followed up and extended to embrace the widely varying conditions found in all coal-mining areas, and that I believe will in time build up dependable information of use to engineers elsewhere. The suggestion that a board of disinterested engineers be designated to continue the studies seems to me highly constructive and one that should be followed out.

The references to the information contained in *Bulletin* 25, which is the report of the conditions under the city of Scranton, have been checked in some respects by German engineers. It was my pleasure to visit some of the German mines in 1922, where the practice of back-filling the mine openings with worthless material and extracting all of the coal has been adopted and very good records of the compression of the material introduced for the support of overburden have been kept.

It is quite gratifying to me because when Mr. Griffiths and I made the tests which are embraced in our report of the supporting value of various classes of material we had nothing upon which to go as we could find no published records of similar tests. However, in actual practice it has since been checked and I would like to see more of that kind of study to determine the values of supporting material.

G. E. Stevenson, Scranton, Pa.—It seems to be my misfortune to disagree with the authors of many of these papers on this subject. I have made an examination of the vein along the line of fracture where pillars had been left that were sufficiently strong to stop the squeeze or cave and in every instance I have been able to get up on the line of fracture of the rock and discover that it is never vertical where the vein is flat but is always inclined upward and outward from the rigid support.

I can give you two or three litigated cases in which that was demonstrated beyond the question of a doubt. One of them is on the line of a street, I think the name is Fiske St., overlying the water tunnel of the Pennsylvania coal tunnel from the Dunmore line down to the Lackawanna River. The pillars, about 35 per cent., were left beneath the street and as near on the side lines of the street as possible. One could travel along that line somewhat irregularly but in every instance where the rock went to the surface the street was adequately supported. There was considerable wash over the line of fracture, which was, as I stated, upward and outward from the rigid support until it reached the wash and then it reversed itself and followed the other line. Of course, the presence of quicksand with its possible opportunity to slide through some crevice into the mine below may materially affect that case.

I have another case in the course of litigation where the rock extends 15 or 20 ft. from the surface. It is under South Main Ave. in Hyde Park, entitled to support because it was a public highway before the coal was severed from the surface. The surface vein is 4 ft. and was mined by longwall on either side of the street but pillars were left beneath it supposedly strong enough to sustain the surface. Wherever the rock approached close enough they did sustain it, but wherever there was 15 or 20 ft. of wash, the same process operated and the same break took place. One can go along the pillars and find that the fracture is upward and outward from the line of support. It was impossible in all instances or many instances to follow it to the wash, but the street is affected wherever we know the wash to be 15 or 20 ft. thick.

G. S. Rice.—How is the longwall approaching it?

G. E. Stevenson.—I am not sure I can tell you that. The only part of it that is accessible now is just along the line of fracture in the mine and between the pillars under the street. The bed is comparatively shallow, less than 100 ft. I am very glad to report that the cave in the lower vein mined first has not materially affected the mining of the solid coal in the overlying vein.

I have now a very difficult problem to settle as to whether or not certain mining in progress beneath one of the most important public buildings of Scranton is going to further damage it by reason of settlement. In 1915 a certain section of Scranton settled some 1, 2 or 3 ft., and this large public building was almost the center of that settlement. At that time the vein we call the New County vein had not been mined. That is a vein about 7 ft. over-all and contains about 6 ft. of coal. A bore hole was sunk into the veins affected and flushing and packing and restoration of the building on the surface were carried on, at a cost of more than $130,000. That began in 1915 and was finished in 1918. We have been constantly watching it since, but no settlement has occurred since 1918. The building is solid and rigid.

When I was first called into the case this surface vein was being mined 60 ft. below the surface, taking out about 50 per cent. of the coal, and except at the edges of the old squeeze which occurred in 1915, the vein was intact. There was scarcely a noticeable crack in the roof or rib, and I made up my mind that mining in that vein would do no harm to the surface provided there was nothing done in the veins below.

That was the New County vein; the next below it is the Clark; then comes the No. 1 Dunmore, which is found in three benches with thin dividers of rock between; then comes the No. 2 Dunmore, which is the vein that collapsed on account of mining a half a century ago, and which caused the trouble in 1915. The mining now starting is in the No. 2 Dunmore vein. The bottom bench, 4 ft. thick, was mined many years ago by water-level mining and very irregular pillars, not uniform in size. In the repairs that were made in 1915 to 1918, rock was packed along the rib and partly filled in the bottom split that had been mined. There are 2 or 2½ ft. of rock; above this, 4 ft. of coal; above that is 3 ft. of coal; above that, 2 or 2½ ft. of rock and then 3 ft. of coal. It is proposed to drop this rock divider into the vein mined many years

ago, lay track on that and take out the 3-ft. bench of coal. When that is completed, the next rock will be cut and dropped and then the top 3 ft. of coal will be cut and taken out.

Wherever this has has been attempted in the vicinity it has shaken things up pretty badly. The question is, shall this be stopped, if possible? This was one of the cases where the owner of the surface has what is known as the third estate. The Supreme Court decided that I may own the right of support over Mr. Conner's coal and under Mr. Rice's house, sort of floating around in the air, and I can quitclaim my interest in it to either one of them and reunite it with their interest, so in that case, Mr. Rice will have a right to absolute support or, if I sell it to Mr. Conner, he will have the right to take the coal out and destroy Mr. Rice's support.

The problem of determining what to do in the matter of support, whether to stop mining or try to condemn the coal or pay the exorbitant price asked, or exercise the right of the third estate, is a difficult one.

G. S. Rice.—Do you not find a very marked difference in the character of the breaks and the surface subsidence as extending either not to the line of the barrier pillar or extending over it according to whether the coal has been extracted from the pillars, or by longwall or semilongwall face; with two conditions, a longwall face that has been approaching the barrier or which is at right angles to the barrier pillar?

R. V. Norris.—There has been little or no longwall experience in the anthracite region, and I doubt whether any anthracite men will be able to answer your question.

G. S. Rice.—In robbing the pillars, do you not find that it makes a difference in ground movement whether slicing them off starts from the barrier pillar, or extraction is begun farther back from the barrier pillar? I have very much doubted, from my observations in studying subsidence, the correctness of hard and fast formulas which some engineers have adopted. Most of these formulas are somewhat theoretical and accept arbitrary values for the strength of rocks. The strength of rock in place is difficult to judge from detached specimens. I find it difficult to accept formulas and strength coefficients of rocks until we have more exact knowledge of behavior of rocks in place, and one must expect wide variations.

H. I. Smith, Washington, D. C.—How deep must the coal beds that are mined underneath be before they interfere with the one above? For instance, in Utah we have a coal bed from 20 to 30 ft. thick, which has a commercial advantage over other coals on account of the breakage of the coal in handling rather than the quality. Above this bed are other beds of coal ranging from 14 to 20 ft. thick. With a coal bed from 20 to 30 ft. do you think it possible to mine the under bed without seriously damaging the other beds 90 or 100 ft. above?

H. W. Montz.—Our experience has been somewhat varied, as I said in the paper. If the vein is first-mined and you rob underneath it, the robbing will certainly damage the upper vein. However, that depends on the thickness and character of the overlying vein. On the other hand, if the veins are virgin and the underlying vein is robbed, subsequent development of the overlying vein will find it distorted, broken up, but the relative location of the vein is the same as originally and it will be found that while it is broken up it can be worked without much trouble. That has been our experience in a number of cases.

I do not think the above would apply to beds 20 or 30 ft. thick; only to veins which have not exceeded, say, 6 ft. In fact, Exhibit I is a vein under which there was considerable robbing done a great many years ago and the vein was subsequently first-mined and robbed, and it was in that section where it had locally settled to the point where the subsidence was 6 in. or so and the main roof was still maintained. In

other words, one could stand in the one breast or one room and look out over the pillars for three or four rooms.

R. V. Norris.—Mr. Montz, if I am not mistaken, in your Exhibit III the seams that were crushed, while they were not fully robbed out, aggregated something like 35 ft. of coal. Have you mined any of the thinner overlying beds since that time, and what was their condition?

H. W. Montz.—In Exhibit III one vein was approximately 10 ft. thick, although it was probably 250 ft. from the nearest underlying vein, which had been squeezed. Two other veins 3½ to 4 ft. thick have been mined over and robbed since that time. We found the contours in these veins very irregular. They did not follow the original contours in the underlying veins at all on account of the irregular movement.

Mr. Stevenson referred to the break extending away from the pillar instead of back over the pillar. That is a perfectly natural conclusion and I think it is the general experience. The roof naturally would break in that fashion and as it reaches its maximum I believe you will find that the break becomes almost vertical. There are a number of instances where the break has been carried in the various veins away from barrier pillars.

I think that what Mr. Rice meant was whether or not there was any difference in the effect of robbing approaching a pillar or retreating from a pillar, and I certainly think there is some difference. It depends largely on what method of mining you use.

R. D. Hall, New York, N.Y.—Was the coal you were discussing last shallow or deep?

H. W. Montz.—It was shallow and deep where the vertical break took place. The top vein, I suppose, was not over 200 ft. in distance.

G. S. Rice.—Where damage was indicated on the surface did the initial breaking occur underground and the break-planes slope back from the solid coal? Was there opportunity to find out whether the break-planes kept the same slope until reaching the surface or whether at some higher elevation the plane of break changed in dip; in other words, whether there was not some horizontal axis where the break reversed its dip and extended over the pillar?

H. W. Montz.—The initial breaks and the ultimate breaks are not the same lines. We have no data in connection with that. At least, I know of none. I have never had occasion to observe it.

Member.—What were the sizes of the barrier in Exhibit II and the thickness of the vein between the coal and the surface?

H. W. Montz.—The barrier pillar was about 150 ft. wide and the vein was 6½ to 7 ft. thick. The wash was approximately 100 ft. and the rock overlying the vein 110 feet.

Member.—Do you mean 150 ft. square or 150 ft. along the line?

H. W. Montz.—One hundred and fifty feet wide. The length is almost indefinite. It would be the limit of the colliery.

S. A. Taylor, Pittsburgh, Pa.—I do not think there is any hard and fast rule as to what interval between seams can be mined without interference of the upper seam. It depends very largely on the character of the intervening strata. I had one particular case where we mined out 6 ft. of coal with 35 ft. of shales above, and there was no interference whatever with a seam of coal 80 ft. above. Yet, in the same Pittsburgh seam a few miles away, we found that the same Pittsburgh seam of coal about a foot thicker had cracked the surface, and there was some subsidence up 400 ft. in elevation above the Pittsburgh seam.

I think the question is one of the character of the intervening strata. If there are shales, they will break and fill up the mined-out space much more quickly than would solid rock or different strata.

A. B. Jessup, Jeddo, Pa.—An outstanding weakness in the many papers and discussions of papers on subsidence which I have read lies in the fact that they did not give a sufficiently complete description of all the circumstances surrounding these interesting occurrences. The nature of the overlying strata all the way to the surface is of the greatest importance. There are a multitude of factors which should be catalogued in every instance if one expects to have a discussion prove of value—the thickness, inclination, nature and make-up of the strata; whether or not they are solid enough to act as a beam or even approximate the behavior of a beam; whether a hard stratum without joints runs through, which would approximate a beam loaded with a lot of loose-jointed masonry on top of it; the span, the location, rigidity and strength of the fulcrums, etc. The more complete the data, the greater the value of the results observed as to subsidence or breaks in predicting the probable action in other cases.

Commercial and operating difficulties in the early days of anthracite mining caused the thickest and best seams of the series to be mined first, leaving numerous thinner or more impure seams supported by the pillars of the thicker seams; or frequently these pillars were drawn, thus causing the overlying seams to subside *en masse.* Better merchandising of anthracite, improvements in the technique of its mining, and especially in the utilization of the smaller steam sizes later, made the thinner seams of commercial value and there arose the problem of their extraction over mining out areas of thicker seams.

In the Wyoming region, where the seams lie comparatively flat, the lowest seam, known as the Red Ash or Dunmore seam, has frequently been completely mined out before the next thin seam above it was touched. This next seam, which lies possibly 60 or 80 ft. above, has been found practically intact after the lower seam had been completely mined out; that is to say, it was in normal position and except for cracks going through the coal and cracks in the roof, very little damage had been done.

An instance of this same condition has been observed in parts of the Lehigh region, where the inclinations range from 30° to 60°, sometimes down to 15°, and since the basin here is of a wide V-shape or more accurately a W-shape, on account of little anticlinals in the middle of it, all sorts of conditions exist for observation as to subsidence. A seam quite variable in thickness but ranging from 2½ to 5 ft. lies from 80 to 100 ft. above the Buck Mountain seam, which is 12 to 14 ft. thick. The Buck Mountain had been mined over the whole area and the pillars had been drawn for about two-thirds of the distance from the surface to the basin when the thinner overlying seam was worked up over the remaining pillars and its workings continued up the pitch to the surface over the completely robbed-out workings below. A line of break was found at the top of the remaining underlying pillars and the overlying vein had slipped down a little at that point. The coal and roof along the line of break were somewhat broken up, but after passing the line of break the conditions of the vein differed very little from those over the area supported by the underlying pillars. The roof was cracked and required more timber, the coal was traversed at intervals by cracks but otherwise no damage had been done at all so far as could be seen. The timbering expense in mining was somewhat increased but other costs were practically unaffected.

As to the inclination of the plane of breaks in the strata, I have traveled along such breaks at the edge of many large pillars and have seen some breaks incline one way and some the other, even at the same pillar sometimes one break might lean backwards and higher up lean up the pitch. I am inclined to think that near the roof the break

in the top rock is more likely to lean down the pitch at the start, and afterward right itself to lean back, if one could examine it all the way up to the surface. Most observations, of course, are necessarily made very close to the top of the vein.

It is only natural that the greatest damage to surface structures or improvements would be along the line of breaks on the surface. The surface damage is usually very much greater where a block of coal is left in directly under overlying improvements for its supposed protection, provided the block is not extensive enough to cause the break which runs up the pitch from cutting through improvements. To protect isolated surface improvements rather than leave small blocks of coal, it is frequently better not to leave any solid but to rob as rapidly, systematically and thoroughly as possible.

If, in every instance of subsidence, all the available facts in relation to the overlying strata, the depth, inclination, etc., are collected and catalogued, a man looking for something to theorize upon, or for a condition similar to the one he is interested in for the possible protection of surface improvements, may look for such conditions as apply to his particular inquiry and then try to apply to it one of the numerous theories. By all means let us get complete facts in relation to instances of subsidence.

H. N. Eavenson, Pittsburgh, Pa.—The evidence obtained so far in bituminous mining practically confirms Mr. Montz's conclusion that the surface of the place robbed is broken mainly over the seam, invariably inside the vertical line.

Prof. Henry Louis has studied subsidence generally, but especially in relation to longwall mining. He says[1] that no matter how one is mining, whether by longwall or room and pillar, there is always a break and it occurs over the solid coal. The records in this country, with very few exceptions, and also the records in South Africa, where they have made very careful studies in some few cases, show that that statement is not correct as far as the evidence goes. He asserts that we are confusing the evidence of subsidence, that we are talking about cracks and placing the subsidence entirely outside of them and that we do not have enough levels to show whether or not there was subsidence.

In some instances he is correct, of course, but I have one case under observation now where the levels extend a long way out of the robbed area. While it will not be ready until next year, I think it shows conclusively that there was no draw at all, that the subsidence was all inside the worked-out area. As far as we know, in room and pillar working that is practically universal. Whether or not it would show subsidence if we had additional levels outside of the breaks and cracks is somewhat difficult to tell.

G. S. Rice.—There are interesting time studies reported in a bulletin of the Bureau of Mines.[2] This study I was instrumental in starting, with H. I. Smith, then in the Bureau of Mines. Subsequently the study was carried on by C. A. Herbert and J. J. Rutledge. There are time studies made in four different districts. One is in the longwall district; it gives the advance on the face at frequent intervals and shows the successive movements downward; the draw was always inclined over the solid unworked coal. The others were made of panels in room and pillar mining and in those cases the break did not reach the vertical; the inclination was backward over the extracted area.

A striking instance of mining in upper beds came to my attention while on a visit under one section of the city of Scranton. The mine was in an upper bed and mining was in progress under the streets when a subsidence occurred in old workings below

[1] Correspondence.

[2] C. A. Herbert and J. J. Rutledge: Subsidence Due to Coal Mining in Illinois U. S. Bur. Mines *Bull.* 238 (1927).

in a deeper bed. The movement extended through the upper bed being mined and surface subsidence covering a city block occurred, reversing the flow of sewage and letting it into the mine through cracks, and also breaking the gas pipes. There was a vertical drop in the street of something like 5 or 6 ft. Mining in the upper bed continued, although with increasing haulage difficulties and admittedly increased cost. However, mining continued until restrained by the city.

R. V. Norris.—I have very little hope that it will ever be possible to prepare formulas that will definitely establish what is going to happen but I believe that with an ample amount of accurate data on record it may be possible to pick out three or four cases similar to the one under consideration and to predict what may happen.

A. W. Hesse, Nemacolin, Pa.—In the mining of anthracite seams, are the two seams mined simultaneously, and if not, at just what intervals of mining are the seams taken out?

R. V. Norris.—They are very frequently mined simultaneously. The second mining is usually one seam at a time.

A. B. Crichton, Johnstown, Pa.—In reply to Mr. Smith's question, I might cite our experience at Portage, Pa. At Portage we mine four seams of coal, one above the other, from the same area. The seams belong to the Lower Productive measures, and are known as Upper Freeport and Lower Freeport, the Upper Kittanning and the Lower Kittanning. Most of the Lower Kittanning seam was mined 40 to 50 years ago over a tract of land of about 5000 acres, and in the early mining few of the pillars were drawn. The interval between the Lower Kittanning and the Upper Kittanning is from 80 to 100 ft., with relatively thick sandstone intervening. Even where pillars were drawn in the Lower Kittanning seam, in the later mining in the Upper Kittanning seam there was no serious trouble experienced, and each seam was mined without any apparent effect upon the other.

Above the Upper Kittanning seam was the Lower Freeport at an interval of 50 to 60 ft. There is quite heavy sandstone intervening. If the Upper Kittanning had been mined out prior to the mining in the Lower Freeport, there was usually no trouble in mining the upper seam, but if mining in the Lower Freeport was carried on at the same time as mining underneath in the Upper Kittanning, there was usually considerable roof trouble in the Lower Freeport—the upper seam.

These different seams of coal were operated under lease by different companies from the same lessor company. After considerable investigation, except in special cases, the engineers of the lessor concluded that it would be best to allow the lessees to mine each seam just as their development proceeded, without regard to the other; that it would be almost impossible to regulate the mining in the lower seam to protect the upper seam in a way that would not cause as much loss as if their mining should proceed in the regular course, or possibly more loss.

Between the Lower Freeport and the Upper Freeport, the interval is 40 to 50 ft., and the intervening stratum is what might be called sandy shale or slate. The mining of the Lower Freeport almost invariably causes trouble in the Upper Freeport overlying, because of vertical breaks. With a heavy sandstone between these seams, I feel the result would be about the same as in the lower seams, as these breaks do not occur in a way to cause serious trouble where the intervening strata consists of sandstone.

H. I. Smith.—What is the thickness of the various beds?

A. B. Crichton.—The Lower Kittanning is 3½ to 4 ft. thick; the Upper Kittanning is from 3½ to 8 ft. thick; the Lower Freeport is 3 to 4½ ft.; and the Upper Freeport is 3½ to 4½ ft. thick.

As to the effect on the surface, that is an entirely different question. We have had mining in the Lower Kittanning seam, where the cover on the coal would vary from 300 to 500 ft. thick, where the mining of the Lower Kittanning seam has seriously affected the surface, causing breaks, and yet has not seriously affected the intervening seams or the mining of the coal in the intervening seams.

This has also been proved by the loss of surface water on a drainage area controlled by the local water company, where the coal 300 to 400 ft. underneath has been mined and the surface broken in such a manner that the drainage area was practically destroyed, except for a short time during the rainy season; the result being that the breaking of the surface allowed the water to reach the mine workings, instead of the surface dam of the water company.

At Meyersdale, Somerset County, Pennsylvania, the Pittsburgh seam of coal, which is about 12 ft. thick, was mined out many years ago. A 4-ft. seam of coal overlying the Pittsburgh seam of coal at an interval of about 30 ft. has been mined out in a fairly successful way. The intervening strata in that case is soft shale. The earlier mining in the Pittsburgh seam did not extract all the pillars, and in the mining of the upper seam now they have considerable roof trouble and caves in the bottom, making more expensive the maintenance of the haulage roads and ventilation. However, they have recovered a good percentage of coal, perhaps as much as is being recovered in the Portage seams, but the mining costs for timber and ventilation are naturally higher due to the disturbance.

R. D. Hall.—The important matter to consider is the nature of the strata with which the mining engineer has to deal. On their character depends the nature of the break that will occur. If this is not remembered, much of the information that would be obtained from the experience in the Wyoming buried valley would be misleading because, on account of the presence of many feet of sand, the roof tends to shear vertically. There are some cases elsewhere perhaps where it shears in steps, so that the subsidence does not appear at the surface to cover as large an area as has been excavated. This has been noted in the Georges Creek field.

Where, however, the roof is solid and thick and there is no heavy weight of sand, it tends to act like a big plate, which in cross-section will resemble a beam. When that condition exists there are breaks back over the coal which have been designated by the mining profession as "draw." I remember that, in 1909, before I had heard of that expression, I made up my mind that what it designated must surely exist, otherwise the roof would not act like a beam. Yet as a beam I was confident it must act. To my surprise I found reference to "draw" in a paper in the *Colliery Guardian*, of London, by a Mr. Dickenson. Though he did not view the failure of the roof as I did, it convinced me that it was true that the roof did act as a plate or beam, and that the right approach to the subject was in this direction. In March, 1910, I published an article on this subject in *Mines and Minerals*, illustrating these breaks and the manner in which they occur.

But there is a condition in roof demolition which causes the failure to vary from that of a beam of homogeneous material. The general theory must view the roof as a monolith, but there is a draw slate below the main roof which does not act as a part of the whole and wherever the stresses are heavy the thickness of the stratum that does not so act may be considerable because of the horizontal shear set up as a result of the bending of the roof mass. This separates the different strata, especially after the roof has broken by tension in the middle of the span.

When the immediate roof has thus sheared from the main roof, the stresses to which it is subject are entirely different, because it is now a built-in beam pinched at the support between the coal pillar and the roof mass above it. It will break as such an *encastré*, corbel or built-in beam may be expected to break. It will not fracture over

the coal. The pinching prevents that. It will break over the excavated area and not far out from the pillar but with a fracture that leans out over the goaf. I have seen these fractures many times. It is not to be assumed that the breaks in the "built-in beam" have anything to do with the break on the surface over the coal designated as the "draw" or with the break in the excavated area at the point of maximum tension of the main roof known as the "main fracture." Why should we assume that the crack on the outside of the mine over the coal is an extension of that inside the mine near the face of the coal, or vice versa? We see only the extremities of the breaks. All that lies beyond those extremities is hidden. Perhaps it is natural to join them up in our imaginations, but seeing that they come of two wholly different stress systems there is no logic that would associate them.

There is a time element in the fall of the roof which cannot be conceived if the roof fails from shear. A roof so failing should rupture suddenly. A five-minute interval of time is the limit of the delay one could anticipate. When the roof fails as a beam it cracks and tries to come down but to do so it must wedge itself into the narrow space between the pillars. When it wedges, an arch action keeps the roof from further failure, but the arch eventually fails by horizontal shear—especially if, by mining, more coal is removed. Then the roof bends and breaks again and as it lowers it wedges again and the arch again asserts itself. And so it goes from beam to arch and arch to beam till the complete failure occurs. When it does, horizontal shear has reduced the roof to a pile of slabs with each stratum resting loosely on the one below. It is this delay that makes it possible to draw out rapidly coal that could not be saved except by such rapid extraction.

J. J. Rutledge, Baltimore, Md.—Mr. Rice referred to Bureau of Mines *Bull.* 358, covering, among others, a longwall coal field where the work was carried on by advancing. This investigative work was carried out under the greatest of difficulties, because we could not name the localities, but so far as draw was concerned, the angle of draw varied between 8° and 9° from the vertical and was always over the solid coal.

F. W. Sperr, Houghton, Mich.—Could you see it?

J. J. Rutledge.—Yes, on the surface; almost parallel to the building, 400 ft. long.

F. W. Sperr.—The fissures extending up, the actual fractures showing?

J. J. Rutledge.—No, we could not see them except at the surface.

F. W. Sperr.—I have made models so I could see the fissures extending upward. (Figs. 8 and 9).

J. J. Rutledge.—We could see the fissures below ground extending above the coal seam for a short distance but we could not follow through to the surface.

F. W. Sperr.—In my models I could see them all the way to the surface.

G. E. Stevenson.—May I ask how much wash there is over that rock?

J. J. Rutledge.—There are shales, limestones, near the coal seam and above these the drift. We have no such wash as you have in the anthracite region. It is a drift, probably 40 ft. thick.

G. S. Rice.—It is a glacial drift.

G. E. Stevenson.—We have a glacial drift in the anthracite region.

J. J. Rutledge.—In Illinois the drift is not exactly of that character.

Answering Mr. Smith's question: As to the mining of a seam above a seam that has been mined below, the Pittsburgh or Big Vein seam was mined probably 10½ or 11 ft. thick with practically no roof over it. There was a thinner coal seam, the Tyson,

FIG. 8.—MODEL EXPERIMENT WITH HORIZONTAL STRATA. DRAW OF FRACTURE TOWARD THE UNDERCUT AREA.

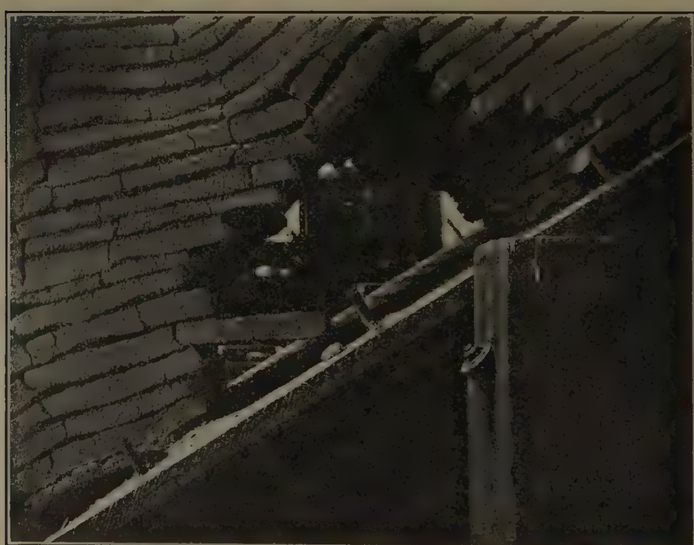


FIG. 9.—MODEL EXPERIMENT WITH INCLINED STRATA. FRACTURES VERTICAL.

90 to 110 ft. above that, with shales and sandstone intervening. It is a very poor roof except for the sandstone, about 10 or 12 ft. above the Big Vein. The Tyson seam is being mined by longwall. There is 100 per cent. extraction, and one mine, at least in the Big Vein, has been pillared below.

Mr. Montz has noted a similar condition. The Tyson, or the upper seam, was practically virgin. The overlying rocks bend but the lower rocks shear where the shales are, as Mr. Crichton said, but the big rock, about 10 ft. above the coal seam, undoubtedly bends.

The most revolutionary development in recent Big Vein mining in Georges Creek has been to take the pillars broadside. Formerly all the pillars were crosscut. Such pillars were, say, 40 to 50 ft. thick, and the coal seam probably 11 ft. high. The former practice was to drive crosscuts, beginning at the inbye end of the pillar, entirely across the pillar; such crosscuts were usually about 12 ft. wide. Two or three men worked in each crosscut, the place being carefully cross-barred as it was being driven. Six feet of solid coal was left in the pillar and then another crosscut of the same size was driven entirely through the pillar. The 6-ft. stump was sometimes mined and sometimes lost.

Quite recently, in one of the mines they have begun to take cuts 50 to 75 ft. wide broadside across the pillars, putting down two, three, and occasionally four or five switches, or turns, leading directly across the pillar. Five men working on each one of these turns make a cut from 50 to 75 ft. in total width entirely across the pillar. Usually a set of five men takes a cut 25 ft. wide then joins on right or left another set of five men taking a similar cut.

H. W. Montz.—You mean in the longwall method?

J. J. Rutledge.—No, I mean in pillar and room work, in robbing the pillars. You take two or three turns, in some cases five, turn into the pillar and, instead of crosscutting, take the full width right across.

H. W. Montz.—There are a number of different methods of mining that have been tried in the anthracite region and I think one of the rather unique ones was the longwall advancing, which was somewhat along the same line although the area had been first-mined. In other words, all of the pillars were mined simultaneously advancing.

J. J. Rutledge.—All of this has been pillared before.

H. W. Montz.—When you are retreating?

J. J. Rutledge.—No, when we are advancing. But the pillar is taken broadside, not crosscut.

A. W. Hesse.—Are two seams being mined in any territory simultaneously?

J. J. Rutledge.—To a certain extent they are now. In earlier years they were mined simultaneously in a few cases. In one mine, many years ago, at Barton, the two seams were mined simultaneously; in fact, they mined the Tyson and Big Vein coal together.

A. W. Hesse.—Are there any data to show the difference in mining cost of the upper seam when mined alone and the cost of same seam when mined simultaneously?

J. J. Rutledge.—There is practically no roof over the Big Vein, while the Tyson had a much better roof, therefore conditions are not comparable.

A. W. Hesse.—Perhaps I did not state my question clearly. In the Tyson seam, that is the upper seam, is there a difference in the cost of production where the Pittsburgh seam has been taken out some time before the Tyson has been attacked?

J. J. Rutledge.—Yes, there is. The Big Vein breaks come up into the Tyson but except where previously there has been some working in the Tyson the Big Vein

breaks do not seriously interfere with the working of the Tyson. They do add some expense.

H. I. Smith.—How much does the break affect the coal?

J. J. Rutledge.—The character of the break does not greatly affect the coal. It shears. As it goes up, it becomes infinitesimal and the shear is not noticeable. Lower down, however, it is noticeable, as in the true longwall method. The upper bed is mined by machine.

T. F. Downing, Jr., Philadelphia, Pa.—There are a number of places in the anthracite fields where the engineers can study this problem very easily. I was particularly interested in the stripping operations where the overburden was being taken from the pillars, where the first mining was done possibly in 1860. I knew of several, particularly in the Shenandoah and Shamokin regions.

Answering Mr. Smith's question, in the Shamokin region, the Mammoth seam, which was 22 ft. thick, took the railroad down from 3 to 4 ft. vertical prior to the Holmes seam being worked, the Holmes being a seam possibly 150 ft. above the Mammoth. The Holmes seam was worked a number of years later. There were times when we were working over the robbed area when the seam broke unevenly and the gangway ran into a piece of rock. Aside from that the condition of the coal seemed to be about the same in that area as it was in the area where it was worked and no robbing had been done.

I think, as Mr. Hall does, that much study must be given to the top conditions, particularly because of safety methods. I know that in the bituminous field many accidents from falls of top are caused by the draw slate breaking back of the coal face.

A. B. Jessup.—Those who wish to get data first hand by observation should go into the Lehigh region of the anthracite fields or into the western middle fields. They would find some splendid instances in strippings where the cover has been removed to a depth sometimes of 125 to 150 ft. There they can make observation in daylight on the action of the strata overlying the Mammoth seam workings (the seam is about 25 to 27 ft. thick) and see the action of the slate overlying it. They can also observe the action of subsidence on a little vein which is about 20 to 30 ft. over the big vein. It is very illuminating and very safe.

First-hand information may also be obtained on these time elements, beam actions, etc., because at one mine there is what is known as a squeeze going on. The robbing has come back to a certain point and the old pillars in a 12 to 18-ft. seam, which were small in this instance, are giving way and overlying strata are coming down in a very complex state. It illustrates the very complex beam action when the great masses overlying the pillars get started. In this particular instance, where perhaps 4 to 5 ft. of the more impure coal on the bottom of the seam was left down, the pillars are very hard and small and are being driven down into it, with the result that the bottom bench is heaving or flowing like pitch, and so far as one can see there is no collapse of the pillars; they are simply being punched down into the bottom coal. The overlying strata and surface are settling gradually without any apparent breaks.

H. Louis, Newcastle-on-Tyne, England (written discussion).—Mr. Eavenson has been good enough to quote some of my views regarding the phenomena of subsidence; he has, of course, only mentioned these briefly, and as he has not represented them quite correctly, it is evident that I have failed to make them clear. Seeing that he, and apparently some of the others who took part in the discussion, appear to hold views different from my own, I think it advisable to put my opinions down as definitely and as clearly as possible, because it appears to me at least possible that some of the differences above referred to may be differences of language and definition rather than of fact. The statements I propose to submit refer to the simplest cases

namely where an area in a practically horizontal bed has been entirely worked out. My experience has been confined to beds of bituminous coal and of ironstone, but I do not at the moment see why there should be any difference in principle between the working of a bed of anthracite and one of bituminous coal, provided that the other conditions are the same. I wish to emphasize that I am considering the conditions of complete working out of the mineral bed with the roof allowed to come down and form goaf in accordance with the general British practice; I am excluding, therefore, methods of hydraulic, pneumatic and hand stowage where extraneous material is brought in to hold up the roof. I am also excluding the case of bord and pillar working where pillars of small size are deliberately left in, apparently with the idea that these will subsequently crush and let the roof down gradually. Such a method of mining has long been discontinued in this country, indeed in Europe generally, and I venture to predict that the time will come when such a recklessly wasteful method will no longer be tolerated even in the United States. Further, I am referring entirely to static conditions after the mineral bed has been worked out and the ground has come to rest; the phenomena that occur during the course of extraction are much more complicated, and these I am not discussing here. Under these conditions, I have hitherto failed to see any difference between longwall working and bord and pillar working when the mineral has been equally completely removed by either method; I do not go so far as to assert that there is no difference whatever between the effects of the two methods, but only to say that if there is such a difference I have hitherto failed to obtain any evidence of it.

When the bed of mineral is thus worked out over a given area up to a solid pillar of the mineral, surface subsidence necessarily occurs. Surface damage may also result, but there is abundant evidence that surface subsidence may occur without surface damage, even to important brick buildings, while it is obviously impossible to have surface damage of the kind we are considering without surface subsidence. Surface subsidence being, therefore, the cause of surface damage, it is all-important to study the former first, and I hold, as stated by Mr. Eavenson, that in many cases the evidence of the two has been confused; evidence as to surface subsidence can only be obtained by careful leveling.

When a portion of a mineral bed is worked out as above indicated, subsidence occurs over the worked-out area and extends, gradually diminishing in amount, until a line of zero subsidence, *i. e.*, a line at which the surface is unaffected, is reached; a plane may be supposed to pass through the line of zero subsidence and the edge of the pillar of unwrought mineral, and the angle which this plane makes with the vertical plane through the edge of the pillar is known as the angle of draw. When the angle of draw extends over the pillar of unwrought mineral as indicated in Fig. 10, I speak of this as positive draw; if the line of zero subsidence is perpendicular above the edge of the unwrought pillar, as in Fig. 11, I should call this a case of zero draw, and if the line of zero subsidence falls within the excavated area and not over the unwrought pillar, as in Fig. 12, I should call this negative draw. When the term draw is used without any qualification, positive draw is always meant, and in my experience I have come across no other. This view appears to be shared by all European mining engineers who have written on the subject, and will be found particularly set forth in the various German publications on the subject and in much detail in recent books by A. H. Goldreich. The view has been very definitely expressed by one of our leading authorities, T. A. O'Donahue, in his evidence before the Royal Commission on Mining Subsidence, where on p. 103 he states "there is a 'pull' or 'draw' beyond the edges of the workings. This has now been well established and is fully recognized," and, further, in a reply to one of my questions (No. 1472, p. 107) he said: "The extent of the subsidence in my opinion is always much greater than the area of working." Since this appears to be the main point upon which Mr. Eavenson and some of the other speakers held views different from mine I am laying considerable emphasis upon it.

I hold that to support a point on the surface a circular pillar would have to be left, the radius of which would be $D \tan a$, where D is the depth of the seam of mineral and a is the angle of draw In practice it is, of course, always an area and not a point that has to be supported, and I hold that the size of the pillar to be left in the seam of mineral in order to support adequately an area on the surface must be equal to that area plus a belt surrounding it of a width equal to $D \tan a$; indeed some authorities speak of such a width as above indicated as the draw, expressing it as a length rather than as an angle. Apparently Mr. Norris appears to hold somewhat similar views inasmuch as he says: "On the other hand, I know of repeated instances where the leaving of a single large pillar for the support of some special property resulted in the absolute destruction of that property from side pull." Evidently what he speaks of as side pull, I speak of as draw, and it is clear to my mind that the reason for the destruction to which he refers is simply that the pillar left is too small. I take it that everyone who has had to do with cases of damage from subsidence due to coal mining has had similar experiences. The evidence concerning draw is perhaps most clearly apparent when considering the pillars which mining engineers are in the habit of leaving for the protection of the mine shafts and of the structures about the shaft mouth. Obviously on account of the draw, such pillars must be larger in lower seams than in upper ones, and as far as my experience goes they are invariably made in this way. When I find mining engineers habitually leaving shaft pillars of smaller size in lower seams than in upper ones, I shall begin to believe in negative draw—but not until then.

The universality of the practice of shaft pillars to which I have referred is, in my opinion, the strongest possible evidence that negative draw does not exist. I believe that the phenomena of subsidence are such as are shown in my Fig. 10, where XY is the edge of the excavation or of the pillar of unwrought mineral, AX being the perpendicular above the edge of the pillar. XB represents the plane of draw; the original surface of the ground (assumed horizontal) is represented by the line DAB and the ultimate result of the subsidence is shown by the line ECB, where C lies within the excavated area. The full subsidence I take, therefore, to be generally over an area less than the excavated area; within this area of full subsidence the motion of the surface is purely vertical and surface damage cannot be done by purely vertical motion; any damage within the area of full subsidence would, therefore, be caused during a preliminary period when that area is in course of being worked out and has not yet been fully worked out. There are numerous examples of even delicate structures within this area going down a good many feet without receiving any damage, and other cases where cracks formed during the course of working have closed again when subsidence was complete. The conditions for little or no surface damage within this area appear to be rapid rate of extraction of the mineral, starting if possible from the center of the structure, and considerable depth of the seam of mineral, or in other words the time occupied in the extraction of the mineral from below the structure in question must be short relative to the time taken by the propagation of the subsidence of the immediate roof of the deposit to the surface. Within the zone CB there is obviously lateral as well as vertical movement; in the lower part of that zone, the stresses are obviously compressive, whereas in the upper part tensile stresses occur; it seems safe to say that surface cracks can be caused only by tensile stresses and that they must therefore be limited to this particular zone.

I have thus given what I hold to be a correct account of the phenomena of surface subsidence that accompany the excavation of a practically horizontal bed of mineral. When the beds are inclined, a much more complicated set of phenomena arises, which I do not propose to discuss here. Further, while I am clear as to the nature of the phenomena, I am unable to quantify any of my data, the figures I have given being purely diagrammatic. All that I am able to say is that as far as my experience goes, the angle of draw in coal measure strata generally varies from 5° to 15°, approaching

the smaller angle the larger the proportion of hard sandstones and the larger the angle the larger the proportion of soft tough shales. In secondary strata the angle of draw appears to be from 15° to 30°. In flexible recent strata such as boulder clay, the angle of draw may be even greater than above indicated, or else such strata may go down bodily, simply molding themselves to the shape of the underlying

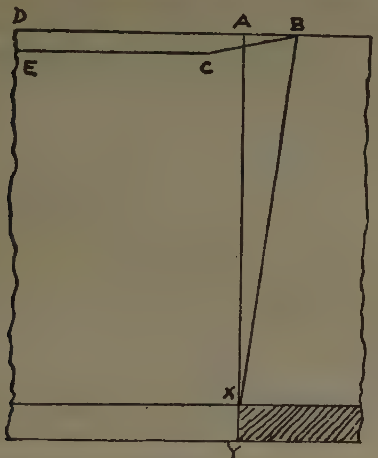


FIG. 10.

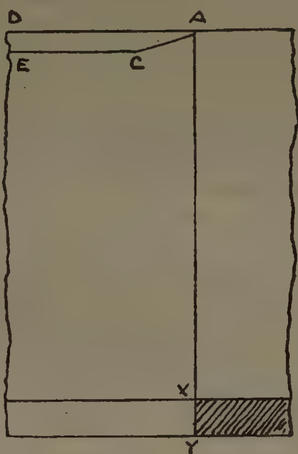


FIG. 11.

harder beds, rounding and softening the angles formed by the latter. The presence of still less consistent beds, such as quicksands, may entirely upset all the conclusions above set forth, and often causes subsidence at points very remote from the excavations that have actually brought it about.

It may be objected that I have confined my observations to the most elementary case, so simple as to be met with but rarely in practice. This is perfectly true, but I hold that until agreement is reached as to what happens in the simplest possible case, it is hopeless to discuss those presenting greater complexities of mathematical analysis.

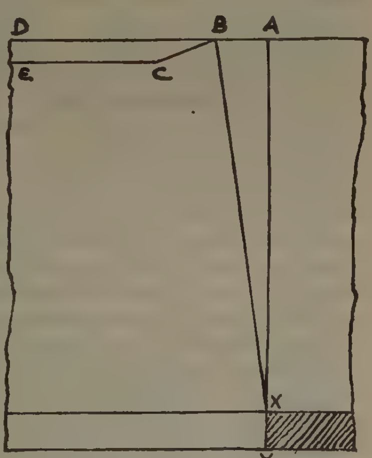


FIG. 12.

H. N. EAVENSON (written discussion).—The writer regrets that he did not represent Professor Louis' views correctly, as he thought that he understood them at the time and tried to explain them correctly. There should be no difference in the action of surface subsidence, caused by the working of any kind of mineral, the methods of working, pitch and thickness of seam and character of strata being the same, and Professor Louis is entirely correct in saying that we should agree upon the fundamentals as developed by the simplest class of cases.

In the United States, unfortunately, there have been few levels taken over areas under which mining was to be done, and nearly all of the records we have where actual levels were taken are of areas covering only the actual mining, and not outside of it, and in such cases of course the information is not such as to definitely settle the point

Professor Louis raises. Among many engineers there has been, I think, confusion between the terms "surface subsidence" and "surface damage," and in most cases when subsidence was referred to, the second term, and not the first one, was meant.

Agreement is general with the statement that in longwall mining the draw is always positive, and all records show that this is true in the United States as well as abroad. In room and pillar mining, in the case supposed by Professor Louis, all the available data were given in the Report of the Subcommittee on Coal Mining to the Committee on Ground Movement and Subsidence.[3] Figs. 27, 29, 30 and 31 show cases of room and pillar mining in India where levels failed to show subsidence outside of the worked-out areas. There are other cases shown where the subsidence found was outside of the area mined out, but in every one there are other conditions, such as an unusual depth of surface wash, irregularity of mining, etc., that obscure the cause of subsidence.

Since the publication of the above mentioned report the record of another case of subsidence has become available. This record shows clearly that there was a positive draw in some cases, but not in all; the level intervals were so great that at times the draw is shown positive where probably it was not so.

It has never been clear to the writer why subsidence over areas mined by room and pillar, particularly where the long break lines of concentrated mining are used, should be different from that over longwall mining, and this record seems to show that in some cases, at least, there is no difference. The writer has persuaded a company operating in Eastern Kentucky to take some levels to test this fact further, and will submit these upon completion, and will also try and secure other data elsewhere.

G. S. Rice (written discussion)[4].—It is evident from Professor Louis' remarks that he holds a different opinion on certain phenomena connected with a specific type of surface subsidence from mining than is held by many engineers in the United States. The question which appears to be at issue is whether or not the line or plane of "draw" produced in ground movement resulting from extraction of pillars may, under certain conditions, be vertical or even inclined toward the workings rather than inclined forward over the unworked mineral, as it is universally agreed occurs in mining by the longwall advancing method.

In discussing this case it is well to take the simplest condition, such as specified by Professor Louis, of a flat bed, whether it be coal, iron or other mineral, where the overburden is stratified rock and where there is no overlying quicksand. Professor Louis says in effect that his experience is that the ultimate angle of draw is always positive and inclined 5° to 15° from the perpendicular over the unwrought mineral. My experience under Illinois coal-mining conditions is that where longwall advancing is used the draw is always positive as Professor Louis describes it, but that in extraction of panels by withdrawal of pillars, the ground movement is manifested, after the first roof beam break, by the forming of an unstable dome over the goaf. Then subsequently, by the falling of individual slabs from the under side, the dome gets larger and higher until it breaks through to the surface of the rock. Up to this time, arch stresses buttress the rocks above the solid coal or mineral. When, however, after the dome breaks through and the central hole enlarges so the arch is no longer sufficiently supported the stratified rocks cantilever out from over the solid unworked mineral and tend to break off in masses triangular in vertical cross-section, until the broken loose material, sliding down against the solid rock, has such a thrust that it balances the tendency for the solid stratified rock to break away and slide toward the excavation.

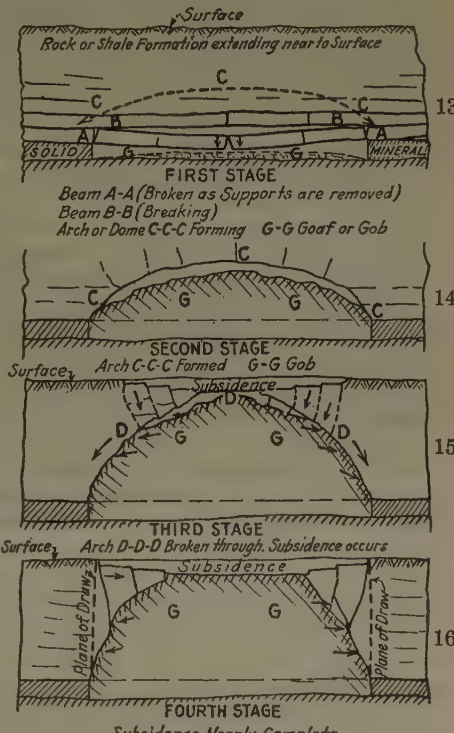
This suggested progressive action is shown by a series of diagrams, Figs. 13 to 16.

---

[3] Report of Subcommittee on Coal Mining to Committee on Ground Movement and Subsidence. *Trans.* A. I. M. E. (1926) **74**, 734.

[4] Discussion of Professor Louis' contribution, for which see p. 138.

I conclude that the final angle of the break or plane of draw is determined by the counterbalancing of several forces, namely, the tendency, through the stress of gravity, for the solid overburden, adjacent to the broken ground, to shear its support diagonally and slide toward the excavation, opposed by the resistance to shear of this support, and the tensile strength in the higher strata against pull or draw, in addition to the opposing thrust of the broken rock buttressed against the plane of the break.



FIGS. 13–16.

The foregoing hypothesis is also illustrated in Fig. 8, 10 and 11 of a paper[5] which I presented before the Institute in 1923, and a photograph of a subsidence dome breaking through to the surface is shown in Fig. 13 of that paper. The photograph was taken from *Bulletin* 17.[6] This work was the product of one of the studies carried on in cooperation by the U. S. Bureau of Mines, the Illinois State Geological Survey and the University of Illinois. Similar findings as regards the movement and subsidence from mining were obtained in a prolonged study, following the establishment of lines of monuments in the respective cases, conducted by the Bureau of Mines over a period of about 10 years under this same cooperation. The results appear in U. S. Bureau of Mines, *Bulletin* 238.[7]

Four conditions were studied under which the draw was determined more or less satisfactory:

1. In the northern Illinois longwall coal field, where longwall advancing is practiced, it was found that the plane of draw was always inclined over the unworked coal; thus giving a "positive draw."

2. In southern Illinois, where a panel room and pillar system was being carried on, a "negative draw" was found. In this case the coal is about 8½ ft. thick, but 1 ft. of roof coal was left in place. The pillar extraction was not complete. A subsidence of about 2 ft., or about 25 per cent. of the thickness of the coal, occurred in the middle of the panel, but did not extend to the sides of the panel, presumably because the remaining pillars and the gradual filling by loose rock of the excavations prevented, or at least has indefinitely postponed, subsidence near the sides of the panel.

3. In southern Illinois, in Macoupin County, where a panel room and pillar system was being carried on, the evidence was incomplete because movements had not yet taken place at the surface. Here the coal is 7½ ft. thick. One-half of the coal was extracted,

---

[5] G. S. Rice: Some Problems in Ground Movement and Subsidence. *Trans.* A. I. M. E. (1923) **69**, 374. See pages 387–391.

[6] L. E. Young: Surface Subsidence in Illinois Resulting from Coal Mining. Ill. Geol. Survey *Bull.* 17 (1916).

[7] C. A. Herbert and J. J. Rutledge: *Op. cit.*

but as the bottom was clay it was expected that there would be a gradual squeeze. At the time the bulletin was prepared, while there was evidence of squeezing underground, the clearance between the roof and floor, which had been 7½ to 8 ft., was only 4 to 4½ ft. at the time the last observations were made.

4. The fourth case was a study of panel room and pillar mining in Franklin County. In this case, as in cases 2 and 3, no pillar of coal was extracted until all the advancing work in the panel had been finished. The coal was 9 ft. thick. One foot was left up as a roof support. The amount of the extraction of the pillars was not obtained. The coal bed is 500 ft. below the surface. Subsequently breaks appeared and a maximum subsidence of 2.67 ft. occurred. There was a slight extension of subsidence at some places beyond the vertical planes extending upward from the sides of the panels, but this I attribute to there being water-bearing glacial gravel and silt 75 ft. thick, overlying the rocks.

In cases which came under my personal observation in Fulton County, Illinois, a 5-ft. coal bed lies in some places only 50 or 60 ft. below the surface and there was no attempt to extract the pillars, which between rooms were quite small, 7 or 8 ft. wide. The roof material was largely shale. The rooms caved as soon as the timbers rotted, or before, and several years later subsidence appeared at the surface. Shearing of the overlying shales was so nearly vertical that on the surface one could see just how the mine had been worked, as the surface practically presented a relief map of the mine in reverse. The influence of geologic faults has, of course, a most important influence on the extent of a draw. I think it is rare when the effects of a draw extend beyond a more or less vertical fault. As indicated in my paper given in 1923, following the extraction of pillars in a panel, domal action would result, and ultimately around the edges of the panel there would be a plane of shear equivalent to a fault or plane, and this would mark the extent to which ground movement would go toward the area over the unworked mineral. Also, the rock over the unworked mineral would stand indefinitely, due to the thrust support of the loose material. If, on the other hand, this loose material were taken away, as is the case in block caving at the Inspiration and Miami mines, then there would be a slide movement toward the area under which the mineral had been worked.

Therefore I conclude, as previously indicated, that while the plane of draw under the conditions named of room and pillar panel withdrawal, the subsidence is generally within the vertical; nevertheless the conditions may be such that the plane of draw will extend over the unworked mineral, as it certainly always does in the case of advancing longwall. It is a question of the balancing of the several forces involved, which are too complicated and variable to express in a general mathematical formula.

C. C. Reid, Cowdenbeath, Scotland (written discussion).—The remarks of Mr. Norris are most interesting and it is evident that he appreciates all the factors governing the subject.

In dealing with pillar and stall work, it is quite clear that it is somewhat dogmatic to figure on any empirical formula on the safe squeezing strength of coal pillars in place since there are no two coal seams with the same characteristics.

An instance may be cited in a coal seam of some 20 ft. in thickness which contains a band of fusain varying between 3 and 5 in. in thickness. In this case, the pillars would be adequate in size for the superincumbent mass under ordinary conditions but it is believed to be the cause of fires on any movement whatsoever taking place in an area which has already been opened out. The pillars are not robbed. In such cases, it is absolutely imperative to prevent any movement whatsoever.

It is not altogether a matter of considering the seams at present being worked but, in many instances, a serious consideration where previous workings on higher or lower horizons have to be treated. Cases are common where, on working new seams, crush has been set up over an already worked area with serious subsidence.

Just as a pillar of support, in an engineering sense, considers height and load distribution, so a pillar of coal must be treated from the points of view of thickness of coal and the superincumbent strata, and since all coal seams are not flat, the further consideration of inclination demands further treatment.

In dealing with the question of flushing or careful packing, the value of which is an extremely variable factor depending on the efficiency of such work, it is quite feasible to visualize that, in a flat working, flushing cannot be nearly so efficient as in an inclined working. Packing must be undertaken before the commencement of subsidence, otherwise an increased subsidence will be the result. Both terms are extremely loosely applied in mining and, unless very careful supervision is undertaken, even hand packing may show very disappointing results.

It is quite evident that the influence of depth has a very important bearing on subsidence and, with the American problem of shallow depths, the danger of surface damage is extremely likely, even with the most careful and systematic packing underground, and an amount of subsidence is certain to take place. While subsidence cannot be prevented, there is no question whatsoever of being able to control it.

In dealing with longwall working as compared to pillar and stall working it is believed that the former system operating on equal thicknesses of seams will show less surface damage provided that adequate packing is undertaken. Longwall work embodies system, regularity and rapidity of operation. It is well known that speed of advance or retreat is an important factor regulating the gradual and continuous subsidence of the superincumbent strata.

Solid pillars of coal abandoned for the purpose of protecting buildings are undoubtedly positions on which fracture has its maximum intensity.

The descriptive work illustrating the facts stated by H. W. Montz are extremely interesting and should form a valuable nucleus for the compilation of data on this important work of subsidence.

Mr. Montz has put his statements forward very frankly and the outline at the conclusion of his paper will no doubt enable mining engineers to formulate information as a guide to future operations.

H. N. Eavenson.—I think that Professor Louis perhaps misunderstood our position. I do not believe that we have yet enough evidence from which to draw definite conclusions, but so far we are safe in saying that the results of our investigations of the records available have shown that there is a considerable difference in the action of room and pillar mining and in longwall mining. Perhaps that is caused by the fact that in most of our cases the records show levels only for the territory immediately over the mined-out area, and that they do not extend far enough away from this area to show whether there was subsidence or not, and it may be that more extended levels will show that conditions are the same as over longwall mining.

I had another check made recently in connection with this matter in a case of subsidence reported by Mr. Hesse, and he said that he did find a very slight subsidence over a solid area but it was only about 50 or 60 ft. away from the worked-out line, the depth being about 500 ft. The levels in this case were taken along a public road, and as a great deal of work has been done on our roads, it is possible that they may not be in exactly the same place, but this is one of the few places where levels show subsidence extending outside of the worked-out areas.

R. D. Hall.—I think that the difference in the point of view between the mining men in England and the United States is due to the fact that in England they, in general, are mining deep seams and here we, usually, are mining shallow seams.

The use of the words "negative draw" is objectionable. "Draw" is the same as "drag," and there is no drag at the surface when the roof breaks between pillars.

When the roof breaks there, it is often due to shear and not to bending moment. The lower measures sometimes break one by one each a little farther from the line of the pillar until occasionally the surface of the ground is reached. These latter actions are due to bending-moment failures of slabs that are detached from each other because of a natural polylithic structure or because of swelling ground.

We have in some places distinct evidences of failure from shear. One occurred in the mines of the Consolidation Coal Co. The measures broke in large vertical and horizontal steps like those of an ancient pyramid from the surface to the coal or, if you will, from the coal to the surface, for the failures were probably synchronous. The engineers measured the break crawling up in the crevices resulting. The cover was shallow or the shear would not have occurred.

There are many instances throughout the country where there have been breaks over the coal with room and pillar mining. Some time ago I received a letter from an engineer in Virginia saying that at his mines there was a break of this kind over an area that had been first mined. It extended 200 ft. over the pillar line which was a long one. The gap made in the surface at the point of draw was a foot or so wide.

G. S. Rice.—Is that in retreating longwall?

R. D. Hall.—No; it is in the removal of room pillars along a long "break line."

G. S. Rice.—That for all intents and purposes is longwall. Where you have a limited area mined by room and pillar work as indicated by survey in subsidence studies by J. J. Rutledge and others in Illinois or in the studies made by H. N. Eavenson, the line of break or draw generally does not extend over the solid, unless there is overlying drift in silt material which runs or slides. The foregoing conclusion is conditioned on flat-lying beds, the generally strong rocks found in coal measures in the United States and mining at moderate depths, say, less than 1500 feet.

R. D. Hall.—Of course, there is never a break up to the surface over a single room unless the bed being worked is relatively shallow.

The Royal Commission on Mining Subsidence

By Henry Louis,* Newcastle-on-Tyne, England

(New York Meeting, February, 1929)

The work performed by the Royal Commission on Mining Subsidence is likely to prove of permanent value, less perhaps for the conclusions it has reached and for the recommendations it has based upon these than for the admirable exposition which it affords of the law concerning mining subsidence and of the legal status of the person who has suffered damage by reason of such subsidence, as also of him whose working has caused the damage. The Commission was appointed by Royal Warrant in the middle of 1925 and delivered its final report in the middle of 1927. The terms of reference to the Commission were as follows:

"To consider the operation of the law relating to the support of the surface of the land, and of buildings or works on or under the surface, by underlying or adjacent minerals; to enquire into the extent and gravity of the damage caused by subsidence owing to the extraction of minerals and the incidence of the resulting liability; and to report what steps should be taken, by legislation or otherwise, to remedy, equitably to all persons concerned, any defects or hardships that may be found to arise in existing conditions."

Report on the Doncaster Area

The Commission submitted an interim report at the end of 1925, which referred solely to one area, usually spoken of as the Doncaster area, and recommended that a Commission of a more technical character be appointed to investigate this particular case. This recommendation was acted upon, and the Commission thus appointed reported early in 1928. Here it is necessary only to indicate the nature of the special problem that had to be solved, the proposed solution being mainly of local interest. The Doncaster area consists of about 330 sq. miles of low-lying land in the southeasterly portion of Yorkshire and adjoining parts of Lincolnshire and Nottinghamshire. No less than 78 per cent. of this land lies below the 25-ft. contour line, and a good deal is even below high-water level. It includes a large tract of land known as Hatfield Chase, swampy ground which has always been a source of trouble on that account, so much so that in 1626 a Dutchman, Cornelius Vermuyden, was called in by King Charles I to devise and execute schemes of drainage for its reclamation. The entire Doncaster area is

* Emeritus Professor of Mining; President of the Institution of Mining Engineers.

drained by certain comparatively sluggish rivers, numerous artificial drains or water courses and some pumping stations. It is underlain by a series of valuable coal seams, the most important and best known of which is the Barnsley seam. In addition to the trouble caused by the general low level of the land, a further difficulty arises in that the seams dip to the eastward in the same direction as the natural flow of drainage of the area. Since these seams were naturally first attacked at their outcrops, the workings proceeding gradually to the dip, the subsidence produced by these workings tilts the surface in the opposite direction to that of the natural drainage. The thickness of the workable seams is estimated as ranging between 5 and 20 ft. and appears but rarely to be under 10 ft., and the subsidence produced by working the coal appears to be about two-thirds of the thickness of the coal wrought. Such an amount of subsidence would under normal conditions be of but trifling importance, but in this low-lying ground, in the circumstances described, it is a very serious matter and the duty laid upon the Commission specially appointed thereto was to suggest suitable remedial measures.

The Existing Law

To return now to the Royal Commission: Evidence as to the existing law was submitted by Mr. D. D. Reid, M. P., of the English Bar, by the Rt. Hon. H. P. MacMillan, K. C., of the Scottish Bar, and by Mr. E. A. Gowers, C. B. (now Sir Ernest Gowers, K. B. E.), Under-Secretary to the Mines Department, and this constitutes a record which will no doubt be frequently referred to in the future. The whole of this has been ably summarized in the Report of the Commission, the summary deriving a highly authoritative character from the legal eminence of the Chairman of the Commission, Lord Blanesburgh, one of the Lords of Appeal. It is pointed out: (1) that the law recognizes a property in unsevered minerals as complete in every respect as that which it recognizes in land itself; (2) that where the surface and the subjacent and adjacent strata are in separate ownerships the surface in its natural state is entitled to support from these strata, this statement being equally true whether the strata in question consist of coal or some other valuable mineral, or not. If withdrawal of support of the surface is even threatened the mineral worker may be restrained by injunction or interdict, and if support has been withdrawn and damage to the surface ensues, the mineral worker, and he alone, is liable to make compensation for such damage to the surface owner. It is also pointed out that the Courts have always been careful to protect the surface owner, but have always recognized the unfettered right of the owner of both the minerals and the surface to part with either one or other separately, making such stipulations as he may please at the time of severance, but they have always held that such grant gives no authority to let

down the surface unless the power so to do be unequivocally set forth in the instrument of severance; this is true equally when the instrument is a grant of the minerals, by so-called lease or otherwise, with reservation of the surface, or when it is a transfer of the surface with a reservation of the minerals.

The Mines Act of 1923

These statements unquestionably embody the law of the land at the time when the Commission was appointed, but the absolute right of the surface to support has since been profoundly modified by the Mines (Working Facilities and Support) Act, 1923, which came into force on Jan. 1, 1924, and of the operation of which we have, therefore, now had five years' experience. This Act, which is repeatedly cited in the Report of the Commission, has a most profound bearing upon the whole question, and, though it passed into law with but little comment, it may fairly be described as one of the most socialistic measures on our statute book. Its effect is that the right of the surface owner to have that surface supported is no longer absolute. The legislature has enacted that the mineral worker may, under conditions laid down by the Act, let down the surface, though he is still liable to make compensation for any damage caused by the subsidence due to his workings. In other words, the remedy of the surface owner is now no longer an injunction to prevent his surface from being lowered, but merely monetary compensation for any damage he may suffer as the result of such subsidence. This Act immensely simplified the work of the Commission, which in the words of the report is "not now concerned in this enquiry with the restoration of a specific property right, but always with the allocation of a pecuniary burden."

Recommendations by the Commission

The Commission considered in this light the cases of the several classes of owners of surface rights that might be affected by mining subsidence, such as public authorities, public utility companies and private owners, and came to the conclusion that it was only in respect of the last named that the present position required any serious modification and then only within certain limitations which the report defines. The Commission specifically limits its concern to the case of the small-house owner, the worker who is of necessity compelled to live near his work and, therefore, on ground liable to subsidence, while he may also be compelled through the necessities of the case to build his house upon a surface from which the right of compensation for damage by subsidence has been definitely taken away at the time of the severance of the minerals and the surface. The Commission recommends that in such cases, subject to certain specified conditions, the right to compensation

for damage shall be accorded to the private owners or occupiers of such existing houses. With regard to future houses, the Commission recommends that the purchaser or lessee of such property which carries no right to compensation for damage done by the working of the minerals shall be distinctly informed of the risk he is running by an indorsement on the conveyance or lease of a definite statement to this effect. Another recommendation is that power shall be given to interested parties to obtain disclosure, subject to proper safeguards, of the plans of collieries whose workings may in the future affect the surface in which such parties are or propose to become interested.

These are the main conclusions and recommendations of the Royal Commission on Mining Subsidence, and it now remains to be seen whether legislation will give effect to any or all of the recommendations. Whether such legislation ensues or not, the Commission has succeeded in succinctly and authoritatively defining the present position; seeing that this valuable result has been attained, it may fairly be claimed that the labors of the Commission have not been in vain.

DISCUSSION

G. S. Rice, Washington, D. C.—This paper by Professor Louis, a member of the recent Royal Commission on Mining Subsidence, was prepared at the request of the A. I. M. E. Committee on Ground Movement and Subsidence. He points out that the result of the Mines (Working Facilities and Support) Act, 1923, has been to modify profoundly the historic British policy in respect to the unqualified right of the surface owner for complete support of the surface in the case of mining operations under it by a second party.

H. G. Moulton, New York, N. Y.—Two years ago Mr. Bosworth, of Denver, presented a paper[1] to the committee on the legal situation in the United States with respect to subsidence caused by mining. The change in policy which has occurred in Great Britain is far-reaching and appears to place mining operations in Great Britain on the basis as respects surface support which Mr. Bosworth contended should prevail in the United States in spite of many legal decisions to the contrary.

G. S. Rice.—Although I assume that the change in policy in Great Britain would not have any significance as a legal precedent in this country, it does suggest how far a country concerned in its mineral resources may go to attain what most public-spirited men in any country will agree is desirable in preventing the loss of a national mineral resource, even though the ownership of that resource is in private hands. There should, of course, be liberal compensations for actual damages paid to the owner of the surface and some governmental control of the methods used. It appears, from what Professor Louis says, that the change of policy in Great Britain was largely an inadvertent one, but it has placed on a good engineering basis the securing of the maximum amount of mineral attainable by efficient mining and it seems like a more sensible basis than the one which now prevails, under which, for example, in one part

[1] R. G. Bosworth: Does the Subsurface Owner Owe an Absolute Duty to Support the Surface? A. I. M. E. *Pamphlet* 1640 (1927). For discussion see A. I. M. E. *Technical Publication* 116 (1928).

of this country, 50 per cent. of the coal has to be left to support the surface. Also, in some instances, large blocks are permanently left for the support of some insignificant buildings.

The situation in Great Britain will now be more nearly parallel, except in close governmental control of mining methods, to that which has prevailed in France from the time of the French kings when concessions were granted by the crown for mining. The owner of the surface does not own the underlying mineral and now the government, in place of the king, grants concessions to an individual or company to minerals underlying certain areas, laid out by the government, without respect to the surface ownership. These concessions pay an annual rental per unit of surficial area. The chief returns to the government, however, are through a share in the net proceeds of the mining operations. As I understand it, the mining company which has obtained the concession must make its own arrangements about obtaining the necessary area of surface from which to conduct the underground operations.

If surface damage ensues in a mine and subsidence occurs as it always does in France, where the complete extraction of the minerals is expected, the company must pay for the actual damage but not for imaginary damages. The extent of damages is determined by a government commission in each respective coal field. I noticed that in certain cities like St. Etienne where there are underlying thick beds of coal, one of them 40 ft. thick, old surface buildings show much cracking and distortion from ancient mining. Now the government is exercising a closer control over mining methods and to assist the commission in determining future degrees of damage, small cement block-coatings are plastered at intervals over characteristic subsidence cracks, and the date marked on each block. Thus if further movement occurs, it will crack the block, and the amount may be measured, then a new block will be placed over the enlarged crack. In going about the older part of the city one will observe many such marking blocks.

In recent mining operations the government has required improved mining methods, using thorough complete packing of all excavations—ramming in the rock, some of which is quarried on the surface. At some mines the hydraulic stowing method is used. As there are no extensive gravel or sand deposits available, as in parts of Germany, a friable rock is quarried and crushed for hydraulic stowing. By these methods surface subsidence is much lessened and as the method used is longwall it sinks evenly and there is comparatively little damage to surface constructions.

Where the surface improvements are important, as in the case of buildings or canals, or where there are questions such as mining under water courses, the government inspection service has the right to state how the method of mining shall be done and the order in which it may be done, although in the actual operations of mining the initiative rests with the mining company. This plan of controlled mining operations in France, in which the government is virtually a partner in the business, appears unique among national industrial systems of different countries of the world. In so far as a visitor may judge, it works out admirably and coal-mining operations, unlike those of most countries generally, have been profitable undertakings. I have an idea that France wishes to conserve its coal resources at the expense of importing one-fourth of its fuel needs.

In this country, the original colonies doubtless inherited from Great Britain the idea that the owner of the surface owned the underlying mineral, unless severed by sale, also the principle that the owner of the surface is entitled to complete support, unless the deed of sale to him specifically stated otherwise.

H. N. Eavenson, Pittsburgh, Pa.—During the past year, while in the Ruhr, in Germany, I was informed that the practice there is for all coal to be removed and all excavations to be back-filled, as required by the mining law. The operator has the

right to remove all the coal, but must pay the surface owner the damages caused by doing so, and these damages are included as an operating item.

G. S. RICE.—In Germany the Department of Mines has an important influence. It has assumed the right—one of the mine owners recently told me that it is now being disputed—to pass on the mining method and to compel stowing of the workings. Probably this control was not disputed under the former imperial government. I recall, in visiting before the war the Krupp coal mines working under the Krupp works, being told that the government would not permit them to mine there, unless they used the hydraulic stowing method. Recently, when I again visited these Krupp mines, they were using a new method of dry mechanical stowing by a machine which had been designed by one of the engineers, and while the result of dry stowing was not as good as hydraulic stowing the subsidence being 20 per cent. as compared with 5 or 10 per cent. with hydraulic stowing, the government was permitting its use.

In other cases in Germany as in the thick coals of Upper Silesia, the mine owners were said to be protesting against the government's making them, in that case, use hydraulic stowing because of the added cost; nevertheless the government was still proceeding on the basis that it had the right to prescribe methods which would prevent what it considers serious damage, either in loss of mineral, or in affecting the surface.

In certain towns and cities in Germany they have been very stringent in the regulation of mining under valuable improvements. I recall that in one town, Zwickau, in mining a 20-ft. bed of coal the mining company could not expose more than a certain area of unsupported roof before they had to hydraulically stow the excavation. This mine was going under an ancient part of the town, under fine old churches and historic buildings which they valued very highly. The work was being done at a very high cost; they were quarrying friable stone, crushing it and transporting it about 10 miles, before it was put into the mine by water. It was reported that the subsidence of the surface was only about 1 or 2 per cent. I was unable to observe any cracking in the Gothic arches of a large fine old church which was being mined under.

In a talk with Professor Louis last summer he said that when Parliament passed this act they really did not know just what they were getting into and added that its effect was not felt for some years.

H. N. EAVENSON.—He said that it passed with very little attention.

G. S. RICE.—Yes, he indicated that when it passed, mining men did not realize its significance so they unexpectedly have had this new method thrust upon them, which I think is very advantageous from a mining and national conservation standpoint.

L. E. YOUNG, Pittsburgh, Pa.—It seems to me that what we are all thinking about at this stage of the discussion is not specifically the engineering or technical aspect of subsidence, but rather a broader social point of view, namely, how we may recover the coal, develop the coal-mining industry and protect the surface.

Mr. McAuliffe has just commented to me upon the fact that in certain of our agricultural regions the surface has been protected, and a much more valuable natural resource, namely the coal, is left in the ground in order to protect this surface. We are all familiar with that problem in certain districts.

Another point of view is that of a large coal company located in a thickly populated neighborhood, where years ago the coal company acquired full right to mine the coal. In the course of time the city has spread out over the area which may be more or less undermined, and immediately the people who have built fine homes begin to worry about their foundations. Possibly all the coal has not been removed, and they attempt in various ways to prevent the removal of the coal.

The making of reservations means, of course, serious interference with the systematic mining of the coal. In many cases damage would result to these houses in case only small reservations were made. A more serious problem is that of the streets and the public utilities which are established over the coal field.

Mr. Rice has referred to the centrifugal machine in Germany. I have endeavored to find out whether it would have any practical application in those parts of the United States where we have 10 to 20 per cent. of waste material being taken outside. The problem is whether or not we can find some way other than hand stowing for placing that material in the cavities underground in a pillar or block system of mining. Those who are operating under physical conditions of that sort would be tremendously interested in finding some way to take rock out of pit cars and stow it underground at a low cost.

E. McAuliffe, Omaha, Neb.—Any movement of social value on the part of society is generally preceded by an educational campaign, sometimes perhaps rather irreverently referred to as propaganda.

There is a definite economic waste taking place in our coal fields today. Perhaps southern Illinois is an example, where seams of coal 10 ft. thick are being mined on a basis as low as a 39 per cent. extraction. That is an extreme case, but it is the result of actual measurements made some years ago. The overlying surface is a yellow clay that never has raised a crop that compensated for the labor expended in producing it.

Another case recently came to my attention in Colorado. Some time ago the Union Pacific Railroad Co. leased a piece of land in an irrigable district to a commercial operator. He was immediately assailed with injunction proceedings leading to interference with his mining, and so we went to the ground to make a study of the situation. We had the owner of the surface with us, and his plea was that at some future time he expected to carry an irrigation system over this land, and that mining would interfere with this project. The subsidence would not exceed 4 ft.; the ground was hilly and rolling and that situation could have been taken care of by following the new contours that would be established by any subsidence that might occur.

This Institute has a valuable opportunity to start a campaign of education leading to a better understanding of what the operator's right is in the matter of removal.

R. D. Hall, New York, N. Y. (written discussion).—The servitude which the owner of the mineral rights owes to the owner of the superincumbent estate received in the United States its first declaration in the decision of Judge Thompson in the case of Jones *v.* Wagner, 66 Pa., 429 (1870); see "The Law Relating to the Mining of Coal in Pennsylvania," by Albert B. Weimer, p. 149–150.

In this decision Judge Thompson said "The right of supports, *ex jure naturae*, which the owner of the soil is entitled to receive from the minerals underneath, has within comparatively a few years, received much attention in the courts in England, and the rule deducible from the cases in all the courts, the House of Lords, Exchequer and Queen's Bench, is that where there is no restriction or contract to the contrary, the subterranean or mining property is subservient to the surface to the extent of sufficient supports to sustain the latter, or in default there is liability to damages by the owners or workers of the former for any inquiry consequent thereon to the latter.

"This is fully supported by Harris *v.* Ryding, 5 M. & W., *supra*, determined at Easter Term, 1839, in the Exchequer; Humphries *v.* Brogden, 1 Eng. & Eq., 251 (1850), in the Queen's Bench before Lord Campbell, C. J. and Patterson, Coleridge and Erle, J. J.

"The whole question was there discussed most learnedly and ably by the Lord Chief Justice and the same result arrived at as had been in the Court of Exchequer,

supra, and in the case of the Earl of Glasgow *v.* The Hurlet Alum Co., House of Lords, 1850, 8 Eng. Law and Eq. 13. There are many other cases referred to in the English courts to the same effect by Rodgers on Mining, page 455, *et seq.* Among them are Robotham *v.* Wilson, 8 H. L. Ca., 348; Pennington *v.* Gallard, 9 Exch. 1 . . .

"We have no case strictly of authority in our books nor do I find any in the books of our sister states . . . The upper and underground estates being several, they are governed by the same maxim which limits the use of property otherwise situated, *sic utere tuo ut alienum non laedas.*"

H. N. Eavenson (written discussion).—The statement of Professor Louis of the present status of the English law about subsidence is very interesting. Our law is the same as the English law was, prior to the Mines (Working Facilities and Support) Act, 1923, but so far apparently no one here has suggested such a change as was made in England by that act.

Our situation is apparently just the reverse of that in England, as in the anthracite subsidence cases in Scranton and vicinity, and in the recent West Virginia case involving the working of an upper seam, the tendency is for surface owners and the owner of an upper seam to try to take away from the mine operator the waiver of surface support which he already has and has paid for, even though most of the anthracite companies are voluntarily sharing the cost of the damage done by their mining.

In the bituminous fields of this country there are not many cases where the new English law would be of benefit to the mine owner, excepting in cases where an upper seam is being worked with the right of support and it is desirable to work a lower seam, where this law would enable the operator of the lower seam to remove all the coal if he could pay the damage done to the mine above. In a recent Pennsylvania case a company mining a thick limestone seam and having the right of support obtained an injunction against a company mining a thin coal seam about 30 ft. below it, proving that the removal of the coal would so damage the stone that it would be unminable. Under this English law the coal company could have continued operations had it been financially able to pay the resulting damages.

It would probably be difficult under our laws to enforce the requirement of the English law that purchasers of property which carries no right to compensation for damage be informed of the risk by indorsements on conveyances of a statement to that effect, although the provision about disclosure of future mining plans could be handled by the state mining departments.

If the owners of the right to support are properly compensated when such support is removed, the writer can see no objection to such a change in the law.

H. Louis (written discussion).—In reference to the comments of Messrs. Rice, Moulton and Eavenson, the report of the Commission as yet has had no legislative result. Whether any of the recommendations of its report will be embodied in any future legislation or not cannot be stated at present. The profound modification of British policy is due not to any action of the Commission, but to the Mines (Working Facilities and Support) Act, 1923, which came into force shortly after the Commission had been appointed and long before the report of the Commission was issued. As stated by Mr. Eavenson and Mr. Rice, mining men did not appear to fully realize its significance when it was first passed. Since then it has attracted a great deal of attention. A full discussion of this Act will be found in the *Transactions* of the Institution of Mining Engineers.[2] There have been already issued two volumes of cases decided by the Railway and Canal Commission under the Act.

---

[2] J. H. Cockburn: The Principles and Operations of the Mines (Working Facilities and Support) Act, 1923, Pt. I. *Trans.* Inst. Min. Engrs. (1924–25) **69**, 435.

I disagree with the statement of Mr. Moulton that the change in policy due to this Act, is that which Mr. Bosworth, in his paper of 1927,[3] contended for. I think that Mr. Bosworth held that there should be a differentiation in the right to let down the surface as between an instrument granting the minerals but reserving the surface and one transferring the surface with a reservation of the minerals. The report of the Royal Commission states definitely that in the British Law, even as modified by the 1923 Act, there is no such differentiation.

[3] R. G. Bosworth: *Op. cit.*

Subsidence in Thick Freeport Coal

By John M. Rayburn,* Pittsburgh, Pa.

(New York Meeting, February, 1930)

When plans were made for the new mine of the Allegheny-Pittsburgh Coal Co., Springdale, Pa., it was decided to operate on the room and pillar system, but not to have any extensive room development, the

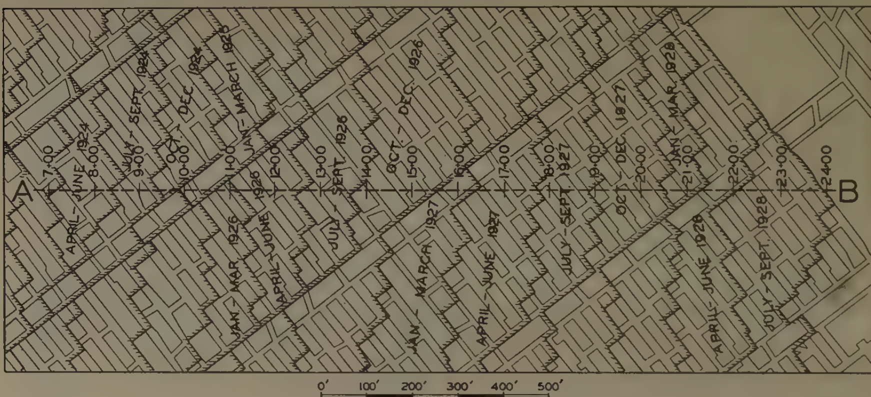


Fig. 1.—No. 1 South rib section.

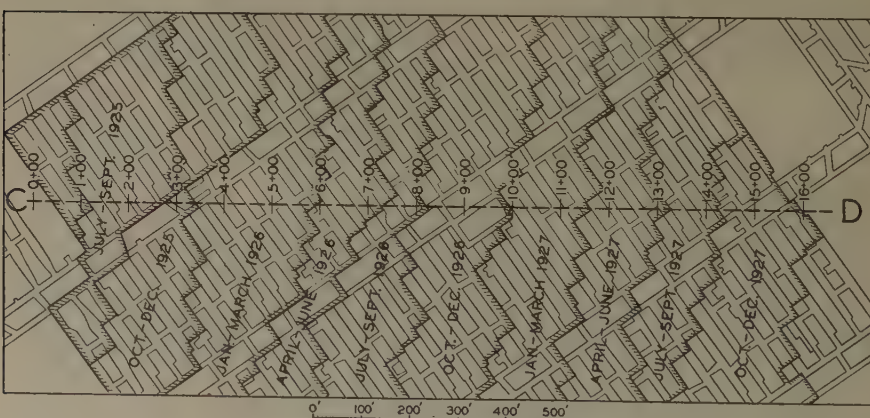


Fig. 2.—No. 2 South rib section.

pillar work being done altogether on the retreat. On each room heading five rooms were to be driven advancing, while five pillars were being removed, so timed as to form a continuous pillar line, with all approaching

* Civil and Mining Engineer.

headings in solid coal. This method gave absolute control over all working places and the percentage of recovery has been unusually large, being over 94 per cent. The vein in the district affected by the subsidence

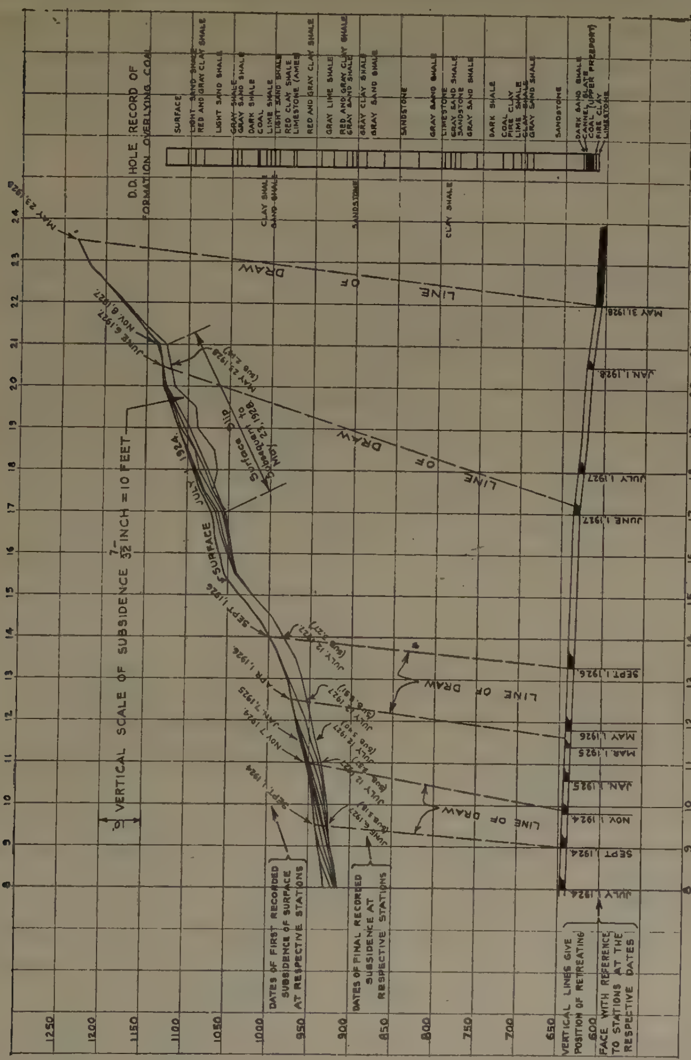


FIG. 3.—SECTION ON LINE AB.

was uniformly 7 ft. thick, 12 in. of which was discarded and gobbed in all rooms in a systematic manner.

The first pillar work in this mine was started in No. 1 South Section, Dec. 1, 1922. As considerable evidence of subsidence had shown up in an adjoining property, the idea was conceived that it would be desirable at this mine to establish base lines parallel to the direction of retreat

or at right angles with the line of pillar extraction or breaking line, and L. W. Cooper, then chief engineer and now general manager of the company, located the lines as shown on the plans. The first line *AB* (Fig. 1) is over the No. 1 South Section and was located July 1, 1924, about 19 months after the pillar work was started. As a result of this, the records for several hundred feet are incomplete, because surface subsidence had already occurred. The second line *CD* (Fig. 2) is over No. 2 South Section and was located at the same time, under similar conditions.

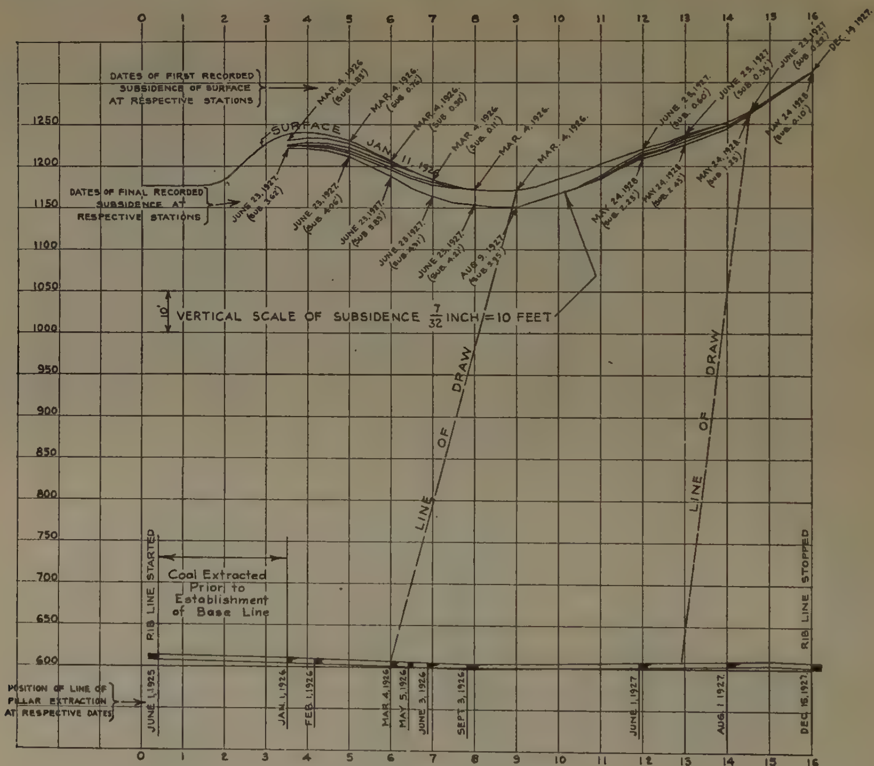


FIG. 4.—SECTION ON LINE *CD*.

Levels have been taken on these lines from time to time and are shown on the profiles, together with a tabulation of the elevations at various points at different dates (Figs. 3 and 4 and Tables 1 and 2).

The record of the diamond-drill hole shows the type of formation overlying the coal. Unfortunately for the complete record, the company has been compelled to cease pillar work in these two sections for some time, but it will be resumed later. The four years shown in this record have apparently covered the greater part of the vertical subsidence;

TABLE 1.—*Record of Surface Subsidence over No. 1 South Rib Section along Line AB*

(Subsidence in Feet)

Station	Sept. 1, 1924	Nov. 7, 1924	Jan. 7, 1925	Mar. 6, 1925	Dec. 1, 1925	Feb. 1, 1926	Apr. 1, 1926	June 1, 1926	Sept. 1, 1926	June 6, 1927	July 12, 1927	Sept. 19, 1927	Nov. 8, 1927	Jan. 10, 1928	May 23, 1928
8 + 00	1.39	2.26	2.54	2.56	2.90	2.95	3.00	3.10							
8 + 50	0.56	1.50	2.16	2.32	2.80	2.95	3.02	3.10	3.12	3.16					
9 + 00	0.07	0.45	1.27	1.63	2.36	2.69	2.88	2.99	3.02	3.10					
9 + 50	0.03	0.16	0.63	1.05	1.59	2.19	2.66	2.97	3.08	3.18					
10 + 00		0.10	0.26	0.51	0.89	1.32	2.12	2.79	3.04	3.17					
10 + 50		0.04	0.14	0.24	0.45	0.71	1.41	2.54	3.11	3.34					
11 + 00		0.02	0.11	0.13	0.25	0.40	0.74	1.89	2.95	3.36	3.37				
11 + 50			0.08	0.08	0.17	0.19	0.30	1.07	2.74	3.40	3.40				
12 + 00					0.06	0.06	0.06	0.34	2.83	3.49	3.50				
12 + 50							0.01	0.09	1.02	3.51	3.51				
13 + 00								0.06	0.42	3.56	3.58				
13 + 50									0.10	3.35	3.38				
14 + 00									0.01	3.25	3.27				
14 + 50										3.25	3.33	3.35	3.43		
15 + 00										2.90	3.07	3.17	3.21		
15 + 50										2.54	2.89	3.14	3.20	3.21	3.24
16 + 00										2.07	2.62	2.97	3.19	3.24	3.27
16 + 50										1.43	2.13	2.97	3.31	3.34	3.44
17 + 00										0.78	1.42	2.59	3.13	3.31	3.40
17 + 50										0.68	1.12	2.39	3.25	3.76	2.85
18 + 00										0.43	0.67	1.54	2.81	3.39	5.45
18 + 50										0.32	0.43	1.94	2.12	3.04	6.34
19 + 00										0.26	0.31	0.55	1.31	2.35	4.61
19 + 50										0.24	0.30	0.51	0.95	1.69	5.69
20 + 00										0.09	0.12	0.16	0.31	0.70	2.68
20 + 50										0.10	0.10	0.13	0.21	0.46	2.14
21 + 00													0.15	0.46	1.74
21 + 50															0.92
22 + 00															0.57
22 + 50															0.58
23 + 00															0.14
23 + 50															0.03

it is likely that subsidence will continue, but in small amount. From the profile it will be noticed that the breaking of the surface has caused rather extensive landslides, which has prevented any determination of lateral movement. This, no doubt, occurs to some extent in most if not all cases.

TABLE 2.—*Record of Surface Subsidence over No. 2 South Rib Section along Line CD*

(Subsidence in Feet)

Station	Mar. 4, 1926	May 5, 1926	July 5, 1926	Sept. 5, 1926	June 23, 1927	Aug. 9, 1927	Oct. 6, 1927	Dec. 14, 1927	May 24, 1928
3 + 50	1.83	2.78	3.12	3.14	3.62				
4 + 00	1.49	2.65	3.14	3.26	3.83				
4 + 50	1.11	2.41	3.01	3.28	4.02				
5 + 00	0.76	1.90	2.71	3.17	4.06				
5 + 50	0.47	1.64	2.43	3.08	4.14				
6 + 00	0.30	0.95	1.77	2.53	3.83				
6 + 50	0.16	0.50	1.28	2.05	4.19				
7 + 00	0.11	0.27	0.89	1.49	4.31				
7 + 50	0.05	0.10	0.26	0.66	4.09				
8 + 00	0.06	0.09	0.30	0.45	4.21				
8 + 50	0.03	0.08	0.08	0.11	4.13	4.16			
9 + 00	0.06	0.06	0.06	0.06	3.91	3.95			
9 + 50					4.04	4.12			
10 + 00					3.78	3.93			
10 + 50					3.55	3.84			
11 + 00					2.90	3.56	3.70	3.75	
11 + 50					1.51	2.17	2.51	2.61	2.66
12 + 00					0.60	1.31	1.89	2.17	2.23
12 + 50					0.48	1.19	1.93	2.39	2.44
13 + 00					0.36	0.94	1.68	2.34	2.43
13 + 50					0.09	0.48	1.08	1.87	1.99
14 + 00					0.23	0.45	0.80	1.62	1.74
14 + 50					0.22	0.33	0.39	1.12	1.23
15 + 00								0.27	0.40
15 + 50								0.08	0.15
16 + 00								0.01	0.10

DISCUSSION

G. S. RICE, Washington, D. C.—The layout of the mine, as shown by the maps, is extraordinarily regular and the pillar retreating has evidently been carried on very systematically with long diagonal break-lines in the two sections of the mine. Since the pillars, judging by the map, have perhaps twice the width of the rooms, the work is virtually longwall retreating.

Perhaps for the first time in this country we have a precise record by surveys of the subsidence in such systematic retreating work. It would be extremely interesting if we could have had a full record from the time the pillar extraction started from the respective barrier pillars, but what information has been obtained is very valuable. Apparently the effect on the overburden is practically identical with that produced by longwall advancing. There is the "draw" in advance of the break-line and this is what Prof. Henry Louis terms a "positive draw." That is, the inclination is beyond the vertical and approximates 10° or more, making due allowance for the surface slopes.

H. N. Eavenson, Pittsburgh, Pa.—The case occurred in the mine of the Allegheny-Pittsburgh Coal Co., which is working in the thick Freeport coal. They established over some of their pillar workings survey monuments along two different lines and obtained the levels on these at different times. Unfortunately the records have not been completed because some question arose about mining rights under a small area of the Pittsburgh coal which is in the top of the hills. I was also a little sorry that the elevations were obtained on single lines instead of being for an area. As you see in the maps, they put the survey lines in the two cases at a 45° angle (horizontal) to the workings and about at right angles to the line of retreat.

The results are notable in that it is one of the first cases where the Committee has definite information that the line of draw extended over the solid coal in room and pillar workings.

As indicated on the profile of one of these lines, there was a big surface slip in May, 1928, which was undoubtedly caused by the movement of the strata below; hence, the apparent subsidence beyond the slip is twice as much as on the more level ground. You will note also that in section *CD* the maximum subsidence was 4.31 ft., or nearly 60 per cent. of the thickness of the seam.

G. S. Rice.—I think that such time-subsidence studies as these are what we need in our ground movement studies. We often get records of total subsidence but without any relation to the relative position of the underground work at the time.

In some instances there have been time studies made. A notable one is that which was carried out in northern Illinois under a cooperation between the Bureau of Mines, the Illinois Geological Survey and the Mining Department of the University of Illinois,[1] by which monuments were established over a piece of ground between two longwall mines and on this ground there was a high school building. In our cooperation it seems to me that this was the most ideal place to select and I think results proved it.

H. I. Smith, Washington, D. C.—In the case of the subsidence study in the northern Illinois longwall working, in the preliminary studies in which I took part, we found that there was actually a raising of the surface ahead of the subsidence rather than a draw. The results were published in the bulletin previously referred to.

G. S. Rice.—I have had occasion to study carefully the subsidence results. After Mr. Smith made the start in the surveying, the work was done by a succession of Bureau engineers in conjunction with the engineer of the mining company. Frankly, the surveys do not seem to harmonize. If one observes the respective tables of subsidences which are given in the bulletin it will be noted that the elevations of the different monuments for any one survey seem to run consistently either above or below a previous survey independent of the position of the mine face. Accordingly the curves in the profiles which appear in the bulletin cannot be drawn through the points of elevation on the different dates but have to be averaged. Hence, I do not think that the rise of surface ahead of the subsidence is well established in this report. I do not argue that such phenomena may not take place. In fact, I think it was rather well demonstrated in the famous Oglesby-Marquette case, but I do not think that the surveys above were made with sufficient precision from which to draw conclusions.

H. I. Smith.—Those parts of the surveys for which I was responsible did show a slight rise.

---

[1] C. A. Herbert and J. J. Rutledge: Subsidence Due to Coal Mining in Illinois. U. S. Bur. Mines *Bull.* 238 (1927).

E. T. Conner, Scranton, Pa.—I suggest that the members compare the subsidence effects given in this paper with those presented in February, 1928, by Harry Montz, mining engineer of the Lehigh Valley Coal Co., in his paper, "Subsidence from Anthracite Mining."[2] In this he cites an instance of pillar extraction and the effect upon the surface, the draw being in advance of the solid face, as represented here.

G. S. Rice.—If I interpreted Mr. Montz' paper correctly, he did not find a draw in advance of the extraction. At least a profile as in Fig. 4 of his paper, shows the subsidence in the central block in what he terms the "main break" with vertical break lines. Then he shows a "subsequent break," with the lines of draw extending over the solid on either side. The angle of the draw through the "rock" is 15° (from the vertical) and through the "wash" is 20°.

As concerns the elevation of the surface in advance of subsidence I have thought that this was due to cantilever effects of large strong masses of rock. Personally I doubt whether it will occur where the overburden is made up of relatively soft shales as is the case in most places in the northern Illinois field, but at Oglesby there was a limestone horizon near the surface which, in my opinion, was responsible for the tilting upward of the surface in advance of subsidence.

---

[2] See page 101.

Bumps in No. 2 Mine, Springhill, Nova Scotia

By Walter Herd,* Glace Bay, Nova Scotia

(New York Meeting, February, 1929)

For the past eight years No. 2 mine of the Cumberland Railway & Coal Co., Springhill, Nova Scotia—a subsidiary of the Dominion Coal Co., Ltd.—has had an unenviable reputation for bumps. As the workings extended in certain directions under a greater cover, these bumps increased in severity and it became apparent that some change in the system of extraction was necessary if the mine was to continue to operate.

In September, 1924, following several fatalities resulting from bumps, the Mines Department of Nova Scotia requested from the U. S. Bureau of Mines the services of George S. Rice, Chief Mining Engineer of the Bureau, to report on the conditions then prevalent in No. 2 mine, Springhill. This request was promptly and courteously granted. Mr. Rice made a full investigation and a comprehensive and helpful report, which is embodied in the 1924 Annual Report of the Department of Public Works and Mines, Nova Scotia.

Prior to 1925, the room and pillar method was employed and in January of that year (there being a cessation of work from March to August due to labor troubles) longwall retreating on the west side, on a limited scale, and in a small section on the east side of the mine was started. The major portion of the east side was not operated on this system till August, 1926, as there was developed a considerable area ready for pillar extraction between an arbitrary boundary and a point up to which bumps of any consequence had not been experienced. In the interval development work for longwall retreating below the 5900-ft. level was vigorously pushed. This longwall retreating system was suggested by Mr. Rice as a possible remedy and was heartily endorsed by the company officials.

This paper describes the conditions and happenings subsequent to Mr. Rice's report and more especially the results with the longwall retreating system of mining under the peculiar circumstances pertaining to mining No. 2 Seam, Springhill, at depth, and records the conclusions which experience with several severe bumps under this mode of extraction would suggest. Except where necessary to make the paper intelligible as an entity any duplication of Mr. Rice's report will be avoided.

---

* Chief Mining Engineer, British Empire Steel Corpn.

Location of Mine

The Springhill coal field is situated in Cumberland County, Nova Scotia, and forms part of the Cumberland coal field of Carboniferous age, stretching from Joggins on the Bay of Fundy, where the seams run under the sea, to the town of Springhill, where the coal measures are thrown up against pre-Carboniferous rocks, in all a distance of 25 miles, and having an average width of 12 miles. On the eastern fringe of this coal field No. 2 mine of the Cumberland Railway & Coal Co. is situated.

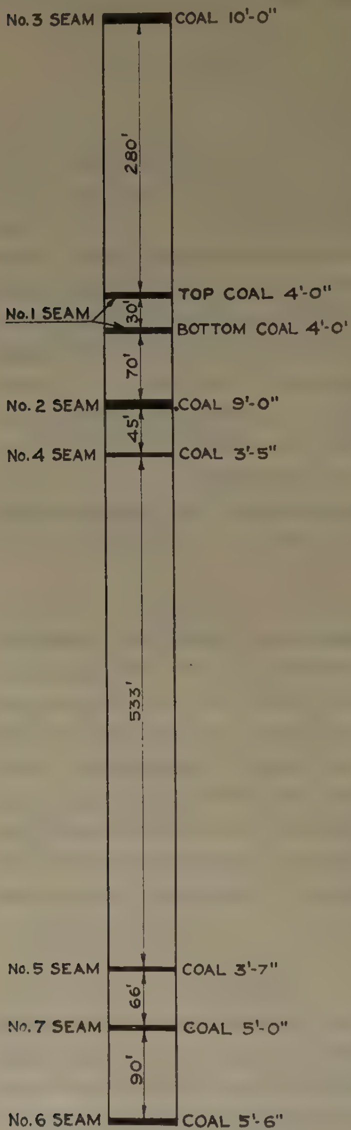


Fig. 1.—General Section Springhill Coal Field.

Seams Mined

In the vicinity of Springhill there are seven workable seams as shown on Fig. 1. These seams are numbered in the order in which they were discovered or opened up. With the exception of seams 4 and 5, all have been mined but at the present time only seams 2, 6 and 7 are worked.

Seam 1, originally 10 ft. thick, split into two seams to the dip and seam 3 workings are flooded, on account of fire.

No. 2 Mine

No. 2 mine consists of three parallel slopes driven on seam 2 from the outcrop a distance of 7700 ft., the slopes being practically on the pitch of the seam. It was opened in 1873 and has produced coal steadily ever since.

Seam

The seam is bituminous coal, averages 9 ft. in thickness, and is free from any dirt or stone partings. It is of medium hardness and has no well-defined cleavage but there are two lines of slips in the coal about at right angles to one another, which however are not continuous in either direction and little difference is noted in the direction the coal is worked. There is a well-defined parting in the seam 14 in.

from the roof. This roof coal is slightly harder than the remainder of the seam.

Inclination

From the surface down to the 2400-ft. level, the pitch averages 30°. At this point the seam is folded vertically downwards for 100 ft., the direction of the fold following the strike. Thereafter the inclination gradually flattens to 20°, which is the pitch in the present workings between 2300 and 2800 ft. of cover.

Nature of Strata

The immediate roof of seam 2 consists of 9 to 16 ft. of strong arenaceous shale and the floor for 9 to 20 ft. is a strong shale of similar character

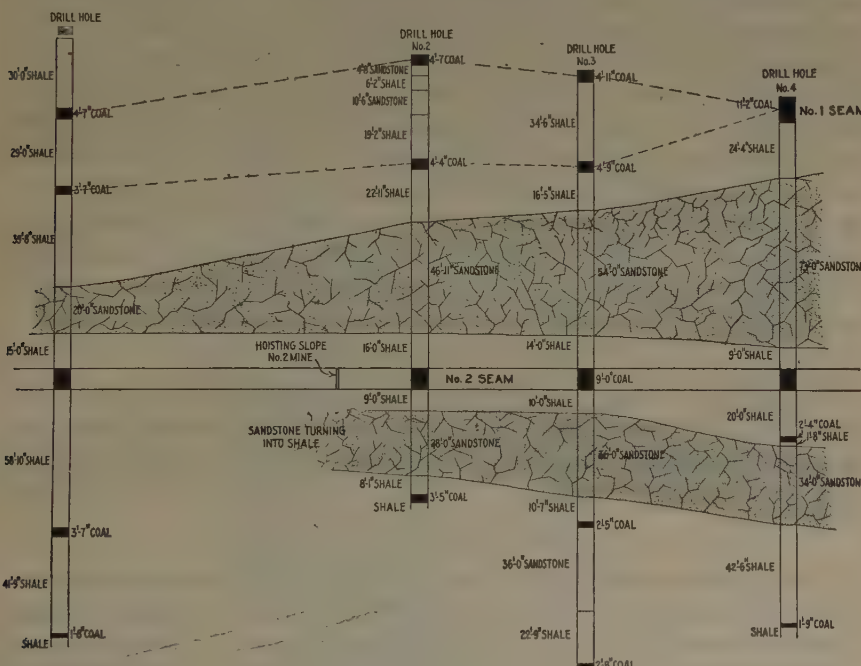


FIG. 2.—SECTION ALONG LINE OF BOREHOLES 1, 2, 3 AND 4 (SEE FIG. 3).

but weaker than the roof. A feature of the coal field is the introduction and disappearance of sandstone bands of considerable thickness within very short distances, often a few hundred feet. This had been noted on the shore cliff 20 miles away but only recently after persistent drilling was a similar condition found to exist and to be a factor in mining seam 2. This is illustrated in Fig. 2, which is a section on a line of boreholes on the strike of the seam, indicated on the mine plan (Fig. 3). Coming

from west to east the overlying sandstone thins from 73 ft. at borehole 4 to 47 ft. at borehole 2, in a distance of 3300 ft., and between boreholes 2 and 1, in 3000 ft., there is a further thinning to 20 ft. At borehole 1 the underlying sandstone is replaced by shale. It is believed the shale roof has considerable elasticity but to what degree has not been determined.

Faults

The field is remarkably free from faults; the few which have been encountered are of little size, a 14-ft. displacement being the maximum. They all run in a true east and west direction (in the line of the dip) as opposed to the fold at the 2400-ft. level, which is on the line of strike. There does not appear to be any undue initial stress in the strata due to faults or folding; the extracted area where bumps have been most severe is a considerable distance away from anticlines, which mark the workable terminations of the seam, as so far proved, in a level course direction. It is admitted there is a greater unrelieved stress in the rocks in a syncline but in this case the synclinal depression is of several miles extent. The change in dip from 30° to 20° is very gradual and drill holes into the upper strata put down from the surface in line with and in advance of the main slope indicate that the dip remains practically constant for another two miles ahead of the lowest workings. In the region where the strata were folded vertically at the 2400-ft. level in the direction of the strike, and in the district where a few faults have been encountered, bumps were not experienced.

Occurrence of Firedamp

Seam 2 gives off gas gradually in the normal course of working but would not be termed a highly gaseous seam. It is not liable to sudden outbursts of gas but because it is dry and dusty shotfiring has been prohibited for many years.

Definition of a Bump

Briefly a bump may be described as the sudden bursting, accompanied by a loud report, of the coal or the strata immediately in contact with it, in contradistinction to the gradual weighting usually accompanying mining at depth. A bump may occur at the coal face, on the roadways leading thereto—particularly if they are in solid coal—or in the waste. The term implies similar phenomena to those described under the heads, "crumps," "quakes," "air blasts," "pressure bursts," "rock bursts," or "rock thrusts."

Description of Bumps in No. 2 Mine

The first appearance of bumps of any severity in No. 2 mine was between the 4000 and 4700-ft. levels at about 1900 ft. of cover, when the system of work was room and pillar (Fig. 3). Three rooms at a time were

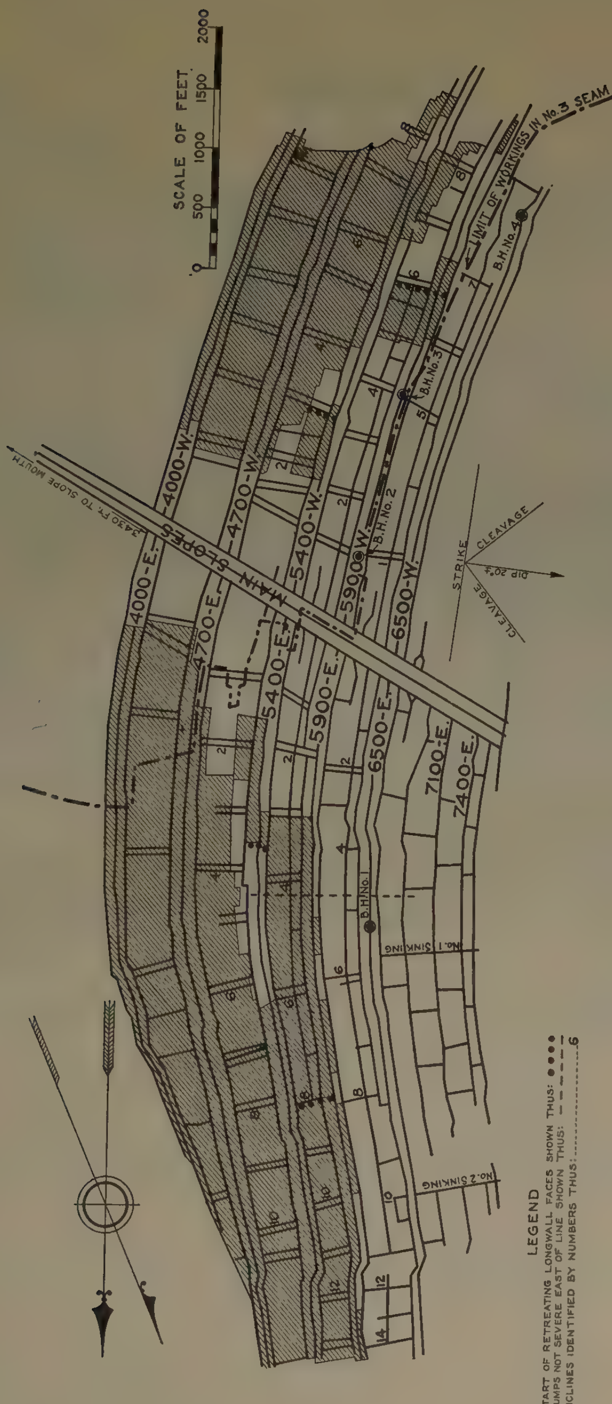


FIG. 3a.—PART PLAN OF No. 2 COLLIERY, SPRINGHILL, N. S.
This plan is a key to Figs. 3b–f.

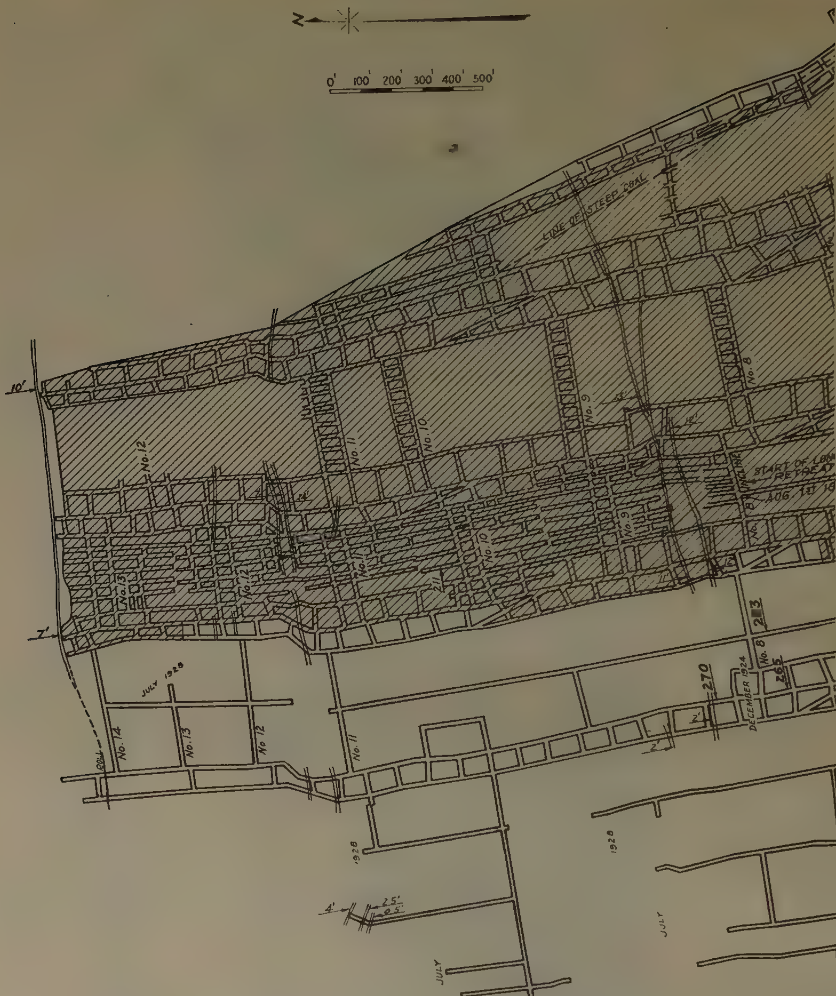


FIG. 3b.—PART PLAN OF No. 2 COLLIERY, SPRINGHILL, N. S.
Note that Figs. 3b–f overlap.

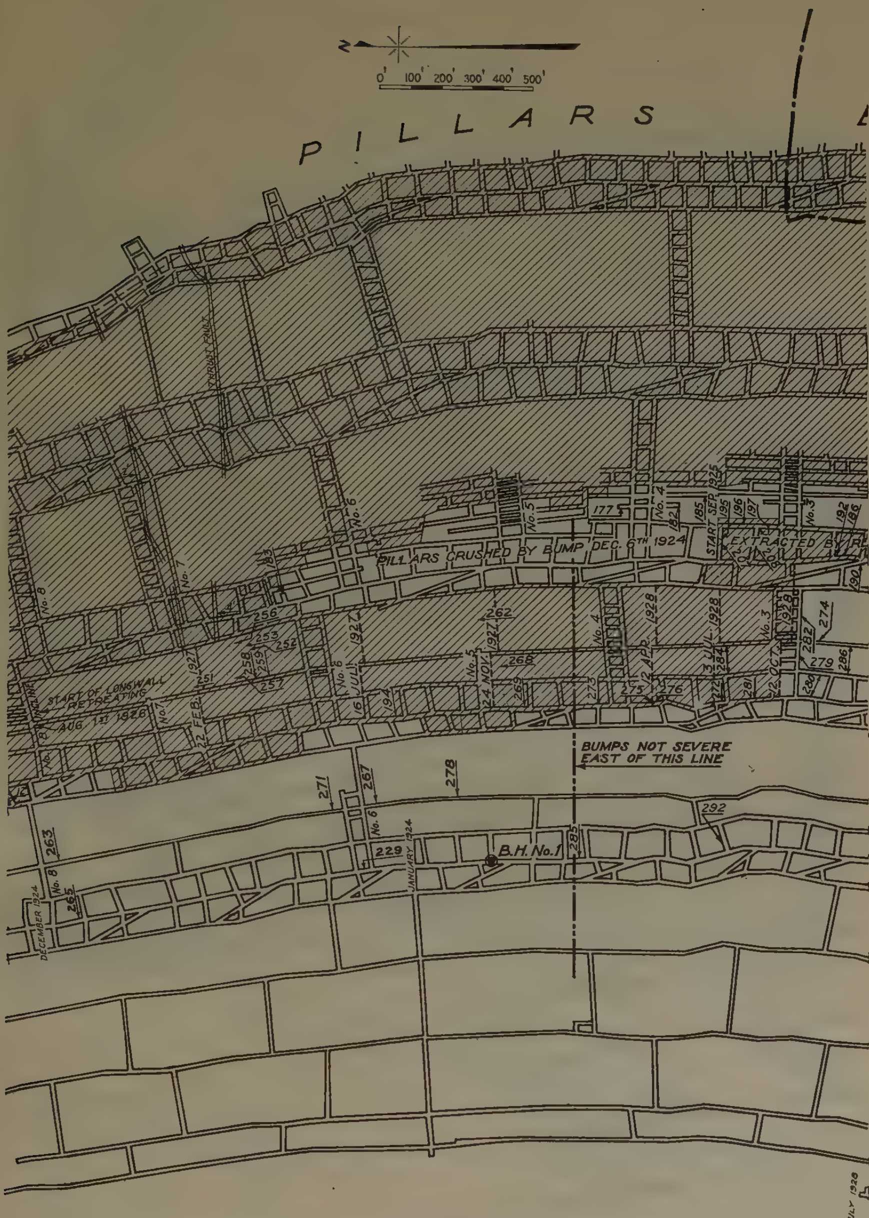


FIG. 3c.—PART PLAN OF No. 2 COLLIERY, SPRINGHILL, N. S.

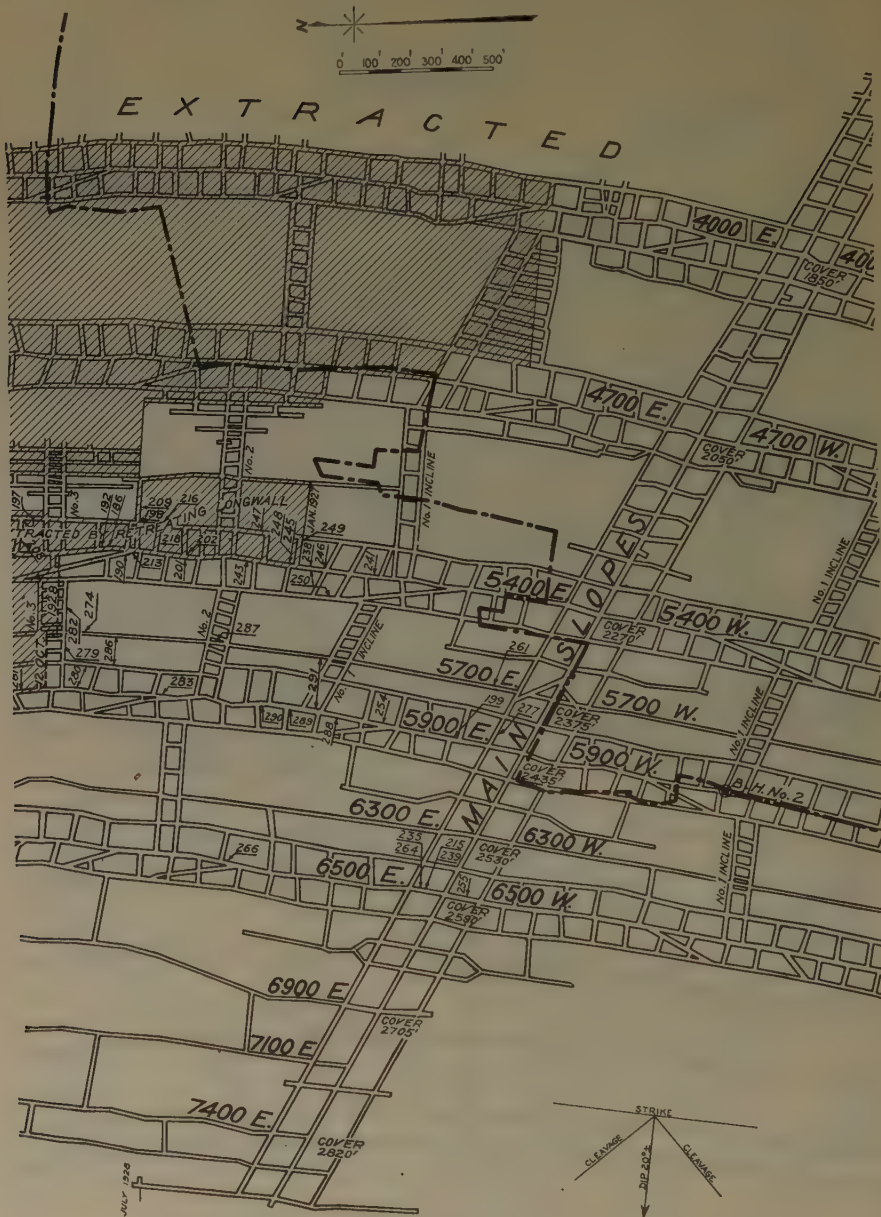


FIG. 3*d*.—PART PLAN OF NO. 2 COLLIERY, SPRINGHILL, N. S.

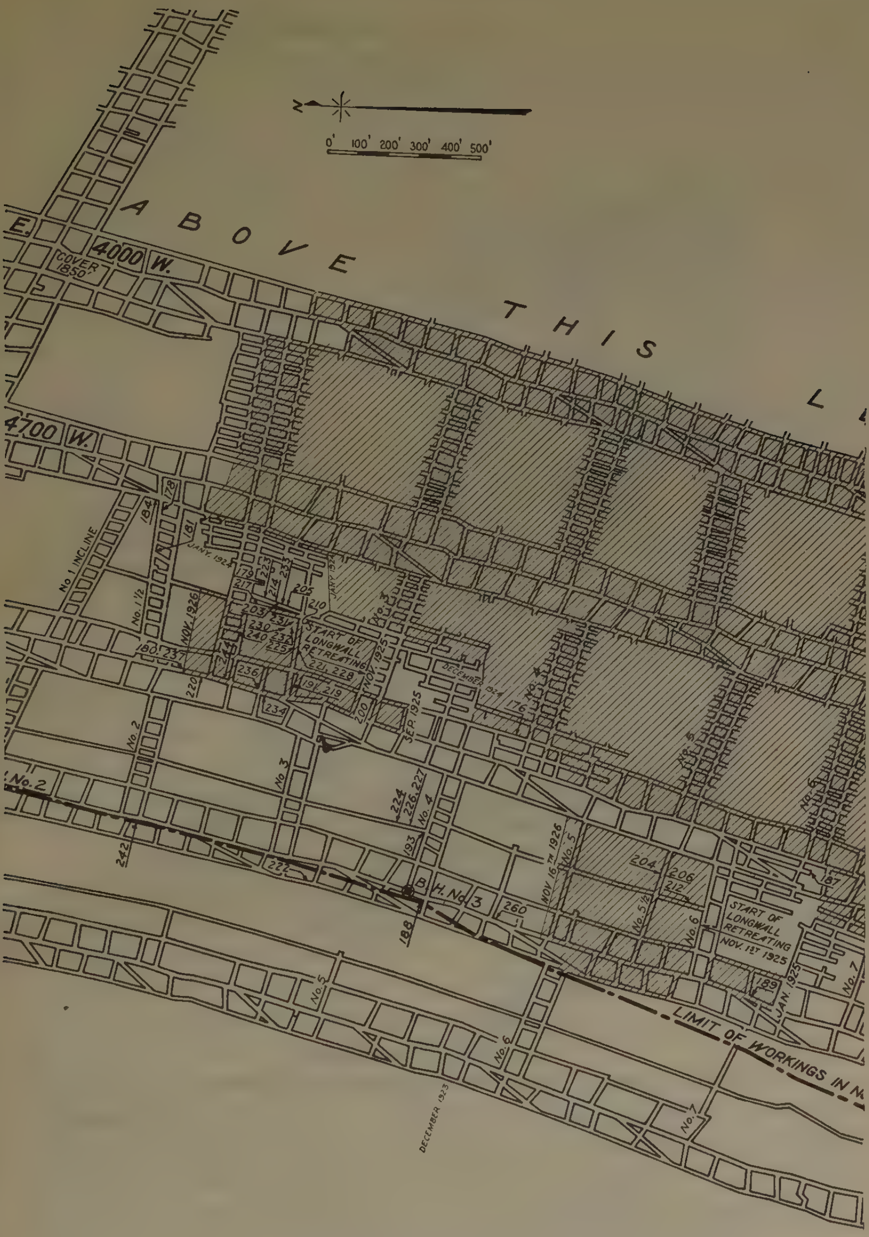


Fig. 3e.—Part plan of No. 2 Colliery, Springhill, N. S.

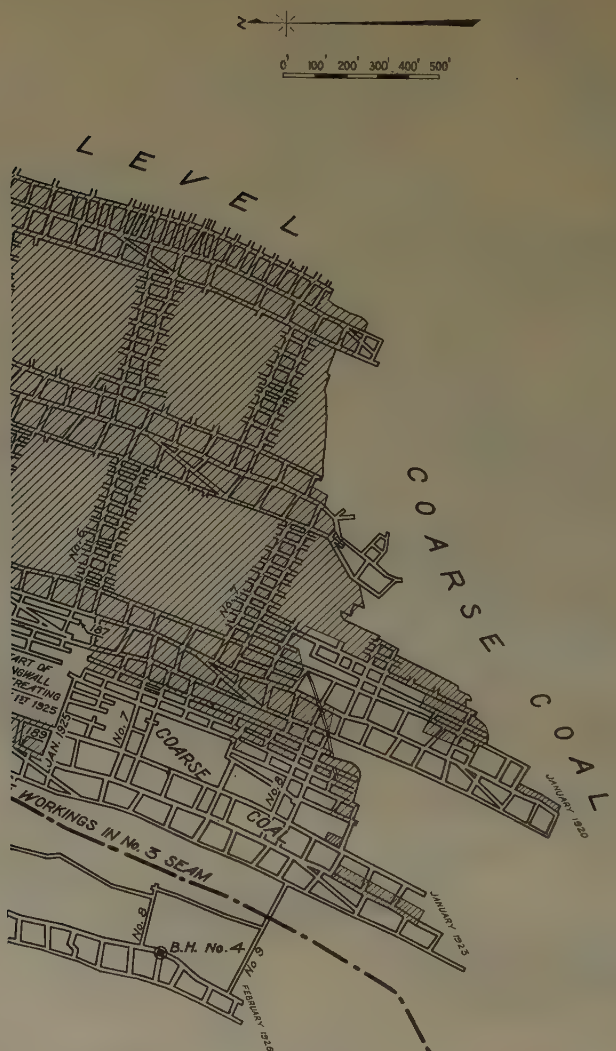


FIG. 3*f*.—PART PLAN OF NO. 2 COLLIERY, SPRINGHILL, N. S.

driven on either side of an incline for 300 ft., on one side to the waste of the inside incline. The pillars measured 35 ft. on the dip and rise and 80 ft. on the strike and rooms and heads were driven 10 ft. wide, but spalling usually increased the width to 14 or 15 ft. at the top. As soon as the top room had reached waste on one side and its predetermined distance on the other, the pillar was drawn and when extraction had continued to the incline another room was started lower down.

To begin with, bumps occurred in driving rooms toward waste, usually when they were parallel to and comparatively near excavated areas to the rise as at the points *A* in Fig. 4. These bumps would occur when the room was anywhere from 300 ft. distant until it was almost through on the waste. When waste was reached and during pillar extraction, a bump never occurred. These bumps caused coal to be suddenly thrown from the low or high side rib, at or very near the working face. The low side bumps, having greater intensity due to the extra resistance encountered, usually threw the track to the high side, but there was no evidence of a break in the floor. It was found at this time in driving a room (level course) parallel to waste to the rise that in many cases the bumps could be transferred from the low to the high side or vice versa by increasing or diminishing the pillar thickness between room and waste.

In driving inclines from a lower to a higher level, to the immediate rise of which the seam had been totally extracted, and in solid coal 500 ft. thick on the dip and rise and often of greater extent in a level course direction, bumps of a similar nature would occur until the remainder was driven through. As the waste in a bumpy district extended, the liability of bumps to occur further from the excavated area increased.

Some time after this type of bump made its appearance and a considerable area of pillars in affected districts had been extracted, waste bumps occurred. These caused practically no damage in the mine and were often unnoticed underground, but caused apprehension on the surface due to earth tremors, shaking buildings a couple of miles from the supposed point of origin and half a mile beyond the outcrop of the seam, showing true earthquake phenomena.

Subsequently, a third type, locally called a "district bump," appeared at 2050 ft. of cover, affecting roadways driven in solid coal, to the dip of totally extracted areas, and extending over a few acres, often closing portions of rooms with coal driven from the low or high side rib and damaging inclines and main levels some considerable distance from the edge of the waste, but rarely affecting the working faces.

District Bump of December, 1924

The largest district bump under the room and pillar system occurred Dec. 6, 1924, in the 5400-ft. east level district at a depth of 2200 ft. It caused a distinct and alarming movement in the town two miles distant

Fig. 4.—District bump of Dec. 6, 1924, 11:40 a. m. Roadways seriously affected are show cross-hatched.

and instantly closed several hundred feet of levels and inclines. A number of men were imprisoned for several hours and one fatality resulted. For the first time the floor was definitely noted to be broken.

Referring again to Fig. 4, which shows the position of the workings at the time of this district bump, the level was completely closed from inclines 3 to 4, the rails being thrown up against the roof and held there by coal and broken pavement. Incline 3 was closed with coal up to the counter level and from the counter level to room 7 heavy falls of stone completely blocked the roadway. The counter level between inclines 2 and 3 was nearly closed with coal thrown off the high side. The top return from inclines 3 to 4, *CD* on Fig. 4, was completely closed, the bottom having been heaved up. Incline 4 was partly closed from the level up to the counter level as the pavement and roof met in the center of the roadway. The pavement broke along the center of the incline and the booms were broken in the center meeting the sleepers. One could crawl up along either rib. The remainder of incline 4 was very badly wrecked. The counter level between inclines 3 and 4 was completely closed but this roadway was almost closed before the bump, by coal spalling off the high side rib. This bump occurred at 11:40 a.m. and at 3:30 p.m. a second bump brought about a fall outside of incline 5, *EF* on the plan. On Dec. 2, a bump had occurred at the face of a room on incline 4, but prior to this the only bump of any severity in this district was in October, also at the face of a room in incline 4.

Following this district bump the driving of all rooms in the vicinity of waste in areas prone to bump was stopped and preparations made to establish extraction by longwall retreating.

The abandoning of the pillars between the waste and main level, which were to some extent shattered by this bump, was at the time deemed prudent, but subsequent experience has proved that a major or district bump never reoccurs in an area which has been previously shaken by a bump and instead of being dangerous, as at one time was thought, the extraction of these pillars would have been a safer operation than similar work in firmer ground.

That district bumps did not occur at lesser depths than 2050 ft. is explained by recent up-drilling to tap the flooded waste in seam 3, revealing the disappearance of the massive overlying sandstone bed to the rise of the 4000-ft. level.

In addition to waste bumps, which occurred some 10 times a year on the average and unfortunately were not recorded as they occasioned no damage in the mine, bumps of the nature first described continued with lessening severity in the vicinity of districts which had been stopped after the 1924 district bump. They seemed to hit haphazard and were mostly from the high side, throwing from a few tons to 20 tons of coal from the upper rib, but occasionally one would occur on the low side. The general

trend was outwards but at times one would occur nearer the waste than a previous one. This outward trend of bumps when mining had ceased was observed by the officials traveling the mine during a strike from March to August, 1925. As they appeared at the time of minor importance, they were not recorded, but it would at least have been of academic interest to know the distance bridged in a given time. They would appear to be caused by the sudden transference of pressure due to the falling of strata far above the immediate roof in worked-out areas.

Longwall Retreating

In January, 1925, a start was made to extract the counter level pillar some 90 ft. thick on the 5900-ft. west section by a straight face parallel to the dip (Fig. 3). This continued for 250 ft. outbye until firmer ground to the rise was reached. (As previously stated in connection with the 5400-ft. east district bump of December, 1924, there was no need to apprehend trouble in extracting the heavily bumped pillars to the rise.) Subsequently two longwall retreating faces 160 and 250 ft. long, respectively, were established. The seam being easy to mine at this depth, handpicks were used in conjunction with shakerface conveyors on a straight face, which experience has proved the best. Similar operations were started on the 5400-ft. level east and west in September, 1925. At a later date, August, 1926, the room and pillar workings on the 5900-ft. level east having reached incline 8, outside of which the harder floor conditions were indicative of bumpy ground, two longwall faces each approximately 220 ft. long were opened out.

The main levels as blocked out for the room and pillar system were about 600 ft. apart, but under the longwall system it was found better to give each face its own haulage on the main and tail system to the main slope, thus necessitating the driving of halfway levels.

The usual difficulties were experienced in establishing these faces with men unaccustomed to this class of work, more especially the drawing of chocks, and before the best system of face support and waste packing with the limited quantity of stone available was found considerable experimenting was necessary. The seam in sections was fully 10 ft. thick and in these trying circumstances the miners made every effort to make a success of this departure from the system they had long been accustomed to.

On the extended faces roof breaks often reaching over the coal occurred about every 40 to 60 ft. of advance and these occasionally closed the face. It was not until nearly 200 ft. of waste had been formed that bumps again appeared in much the same form as heretofore, showing sometimes in roadways previously driven in solid coal and occasionally on the wall faces. A few of these latter had the typical characteristics of a low side bump in a roadway driven in solid coal—conveyor pans and

chocks were thrown from the face towards the waste, the chocks usually being found intact in their new position.

An exception to the above was a bump on the wall face of the 5400-ft. east section with only 30 or 40 ft. of waste formed. This, however, cannot be taken as typical because the ground in the immediate vicinity had been affected by a district bump and the excavation in firmer ground probably caused a further sudden readjustment of strata over a considerable area.

Inbreak of Water on West Side of Mine

In November, 1926, an inbreak of water occurred in the waste of the 5900-ft. west longwall section. There is little doubt that this water came through a break in the overhead strata connecting to the flooded waste of seam 3, some 400 ft. above. The inflow was 130 gal. per min. which has remained constant ever since. Seam 2 had been totally extracted on the room and pillar system under this flooded waste for 10 years previously without mishap, but the thickening of the overlying sandstone beds to the dip and leaving in a row of large crushed pillars between the 5900-ft. west longwall and the upper waste on the 5400-ft. west level (Fig. 3) may have been a contributing cause, together with a heavy waste bump felt in the town a few hours prior to the inbreak.

Considering the violent movements periodically taking place in No. 2 mine, it was considered inadvisable to risk a possible increase in the flow by further mining and in addition to all the west side workings the 5400-ft. east longwall was stopped a short distance before its allotted boundary, until seam 3, holding some 200,000,000 gal., is dewatered, for which preparation is taking place at the present time by upward drilling from seam 2, the borehole connecting to the lodgment of main pumping station in No. 2 mine.

Face Supports

To make the working conditions at the coal face more secure the waste packs were gradually strengthened and better built. After various trials the size and spacing of stone packs and face chocks as illustrated in Fig. 5 have proved fairly satisfactory and since its establishment over a year ago, face bumps have not reoccurred to date. Even the strengthened packs may seem very slight when the height of seam and depth of cover is considered, but it must be remembered that the immediate roof is of exceptional character. The roof grinds on these packs, making good building stone, and rarely breaks down across the walls closer than 10 to 12 ft. from the face, provided the packs are kept well forward. To stow the waste completely would be economically impracticable. Heavy timbers under steel straps were tried in place of hardwood chocks but proved more costly.

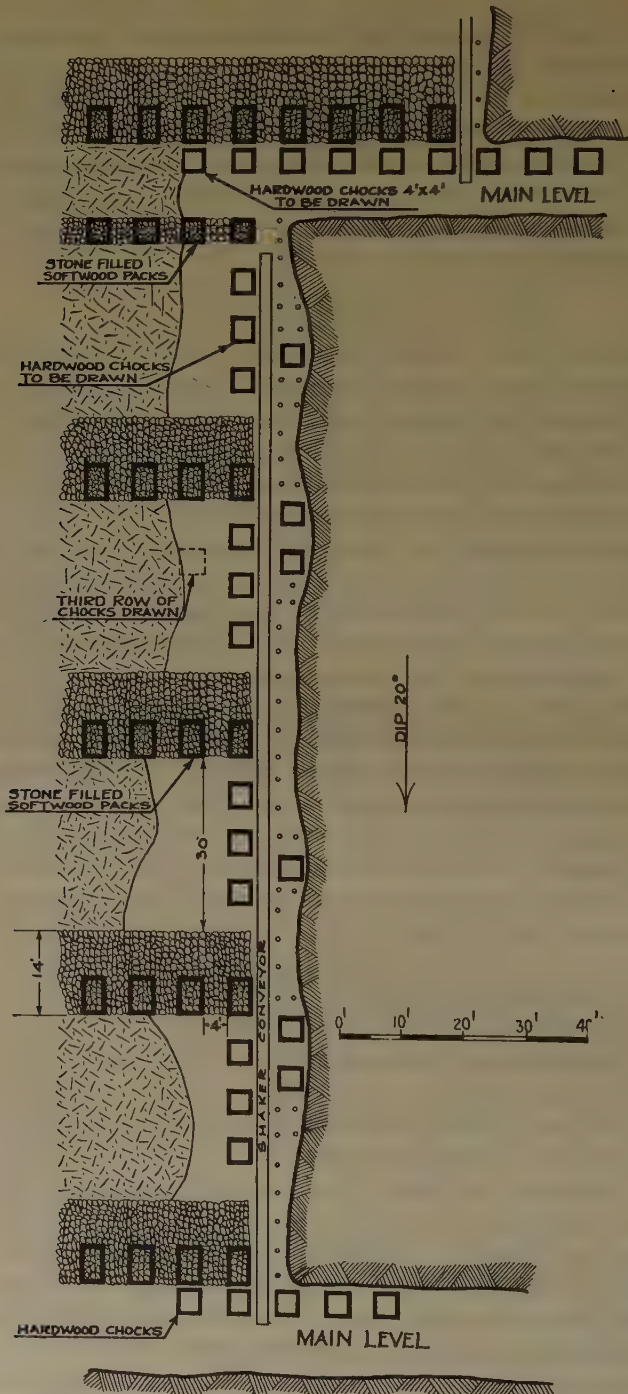


FIG. 5.—SYSTEM OF ROOF SUPPORT ON LONGWALL FACES.

Hardwood chocks made of sticks 6 by 8 in. Stone-filled softwood packs made of sticks 6 by 8 and 6 by 4 inches.

The wall faces are worked double shift and make an advance of 2½ ft. per shift or 5 ft. per day, chock drawing and pan moving occupying the third shift.

The longwall workings on the west side of the mine are not of sufficient extent to prove very much other than that bumps of limited extent could occur on the wall faces under certain conditions of poor waste pack building, but prior to stopping there was dawning the fact that the haulage roads outside the retreating wall faces and for some 30 to 60 ft. distant therefrom were liable to either high or low side bumps at intervals varying from 30 to 60 ft. of face advance, but these did not synchronize with the face weightings which occurred at almost similar intervals.

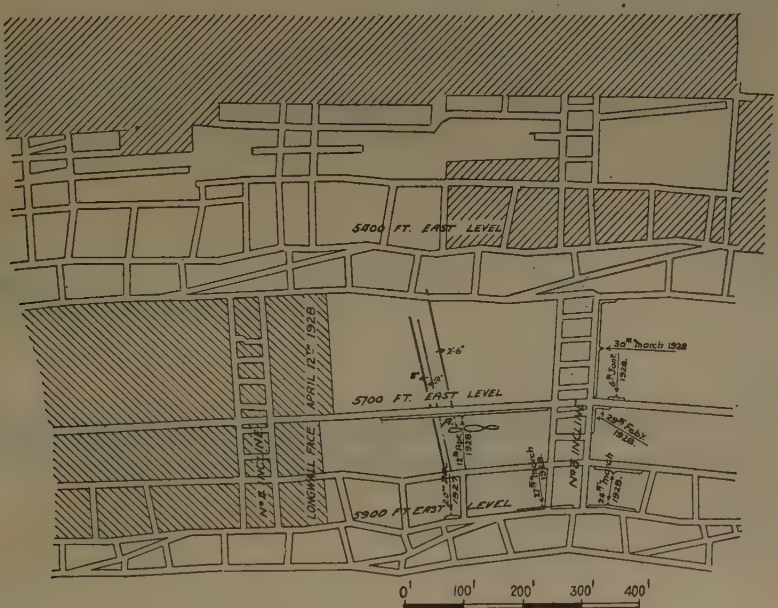


Fig. 6.—Bumps of April 12, 1928.

Arrows point in direction of manifestation of bump forces; brackets indicate areas affected.

For a year after the west side was stopped the east side workings on 5700 and 5900-ft. levels were in an area less liable to heavy bumps, but for the past year the conditions being similar to the west side the same tendencies have been observed, beginning by levels being affected 30 ft. from the wall faces and as these advanced the succeeding bumps striking further ahead than their predecessors. These bumps were of the first-mentioned type, similar to those experienced in driving rooms towards waste.

April, 1928, *District Bump*

On April 12 of this year, a district bump occurred on the 5700-ft. east level at 2500 ft. of cover, the exhibition of greatest force centering some

250 ft. outbye from wall faces. Fig. 6 shows the position of the wall faces at this time in relation to surrounding workings and the locations and dates of previous bumps in the district. Fig. 7 is a larger scale plan showing the effects of this bump on the roadways as measured at the time, and probably its most interesting feature is the location, size and direction of the cavity in the low side pillar, beginning at a point 215 ft. from the face at time of bump and extending laterally for 90 ft. At its maximum it measured 9 ft. across with a depth of 18 in. Its long axis

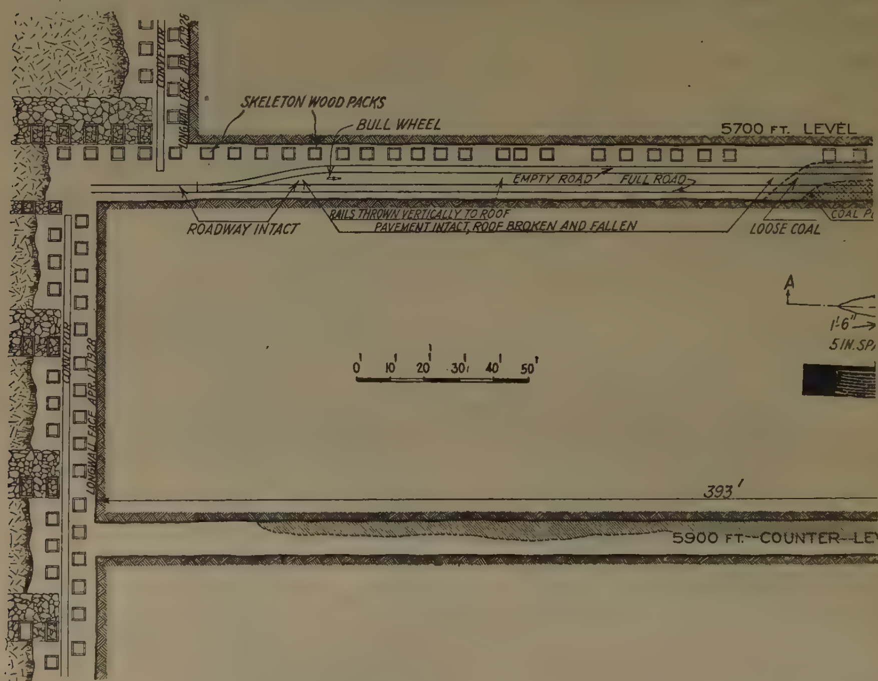
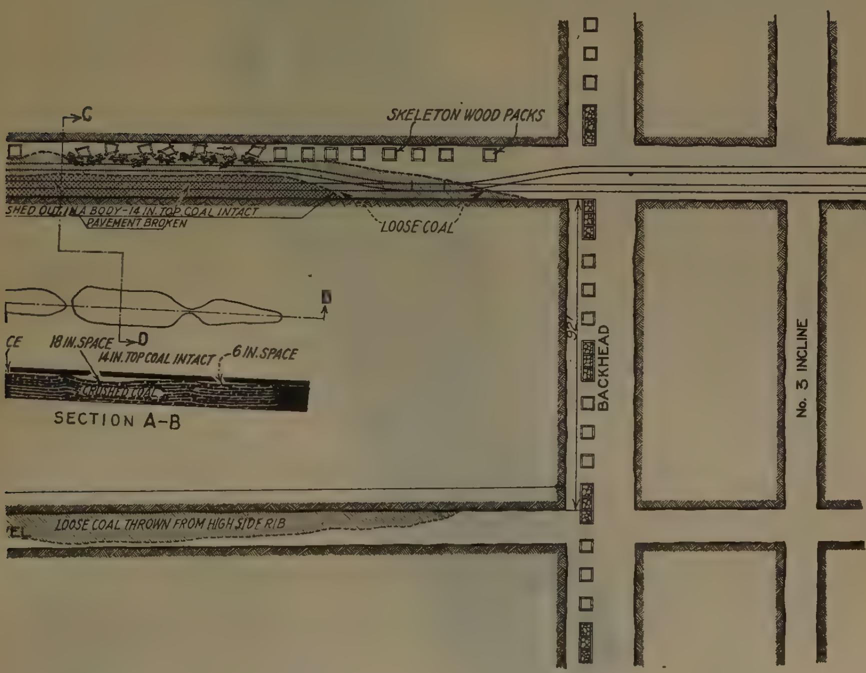


FIG. 7.—PLAN SHOWING EFFECTS

is 35 ft. from the low side of the 5700-ft. level, in a pillar about 400 by 100 ft. in area. It will be noted that this cavity is not continuous but in three sections, denoting a wave effect, and the long axis is at right angles to the wall face. This might be explained by the possibility of an overthrow of weight from the worked-out sections above causing a line of preinduced weakness in this direction or simply that as the pillar was thinner and consequently weaker on the dip and rise it was easier for the blow to split it in the line of strike, and the relative hardness of the roof at any given point is a factor which cannot be ignored.

Fig. 8 is a section across the 5700-ft. level taken immediately after the bump and across the cavity and crushed zone some four months afterwards when it was uncovered as the face advanced. When mining across this

cavity it was observed that the roof and top 14 in. of coal were intact and not different from other sections of the face but that the coal below it and on either side of its center line for 7 or 8 ft. was crushed in lines as shown in Fig. 8, and perceptibly warm; further, that the floor had broken on the center line of this hole and pieces of floor stone had been injected upward into the crushed coal 2 ft. above the pavement, clearly demonstrating that the reaction took place through the floor, the cavity being formed when the floor subsided. The roof movement must have been



OF BUMP OF APRIL 12, 1928.

of very small extent, as props thrown out in level were not broken but hurled to the high side and in an inbye and outbye direction on the center line of the short axis of the central cavity. The floor in the center of the level was broken opposite and for a distance equal to the length of the cavity. That the cavity did not extend to incline 3 may be explained by the fact that the ground in that vicinity offered no resistance as it had been shaken by previous bumps.

It will be observed that the pillar between the cavity and high side rib had been pushed up in a solid body, the coal probably moving with the floor and sliding on a good parting in the seam 14 in. from the roof. In recovering the level the extruded portion of the pillar had to be sheared off and was found as firm as mining in a room. A heavy air blast accom-

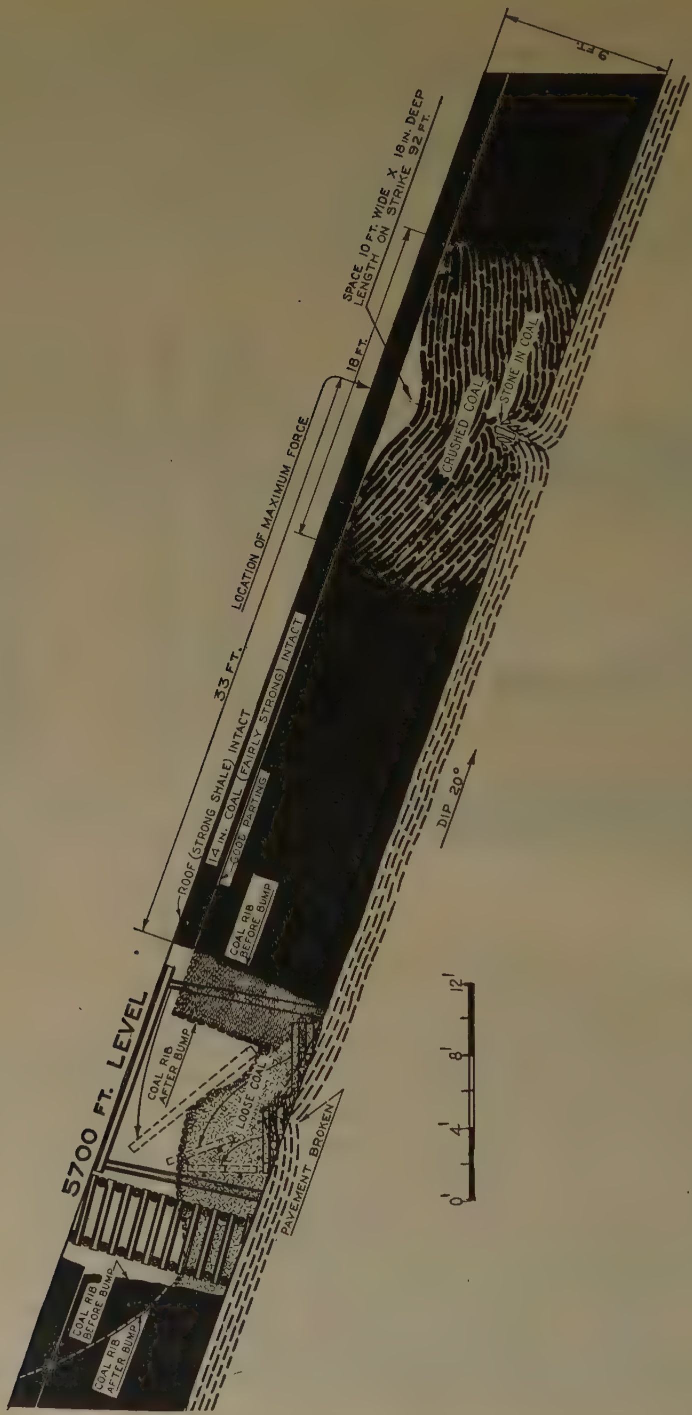


FIG. 8.—CAVITY IN COAL RESULTING FROM BUMP OF APRIL 12, 1928 (SECTION ON LINE *C-D*, SEE FIG. 7).

panied the bump and would be looked for with the sudden protrusion of 500 tons of coal into the roadways. A considerable volume of gas given off could not have been even a contributing cause as it was no more than would be expected from the instantaneous mining of this quantity of coal. Fortunately the bump occurred between shifts, but the wall faces were unaffected, not a stick of timber being out of place.

July, 1928, *District Bump*

On resumption of work after the April bump the district was quiet during a face advance of 200 ft., when on July 3, a second district bump

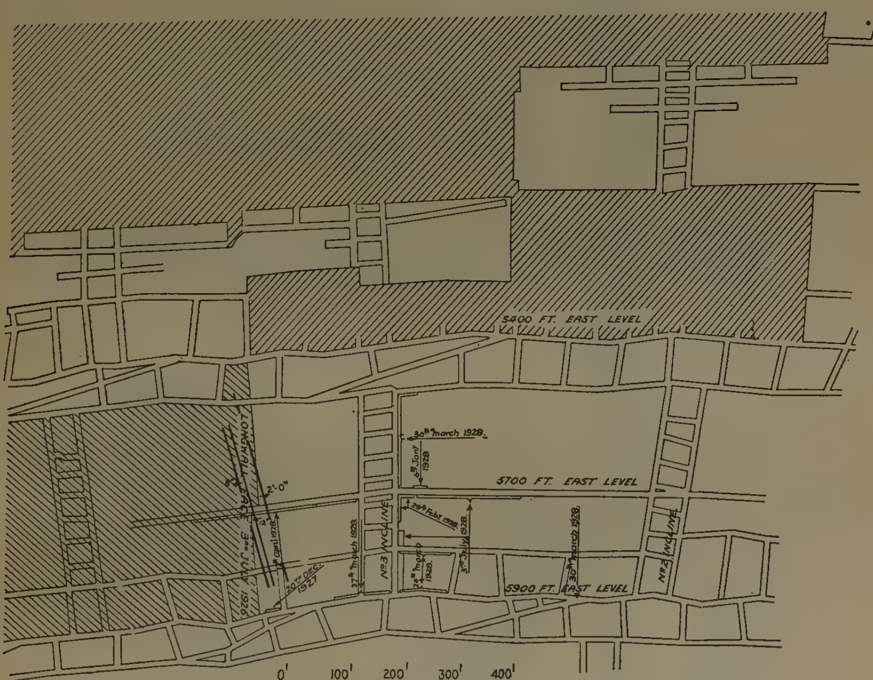


Fig. 9.—Bump of July 3, 1928.

Arrows point in direction of manifestations of bump forces; brackets indicate areas affected.

on the 5700-ft. level occurred with the same characteristics as the previous one—coal pushed up in a body, floor broken and track thrown in a similar manner—only the maximum exhibition of force had extended to 380 ft. outside the wall faces. Fig. 9 shows the position of the face at time of bump in relation to surrounding workings and previous bumps recorded in the vicinity. Fig. 10, on a larger scale, shows the damage to the 5700-ft. level. Unfortunately in this case the trip riders were caught, one fatally injured and the other shocked and gassed. The air blast

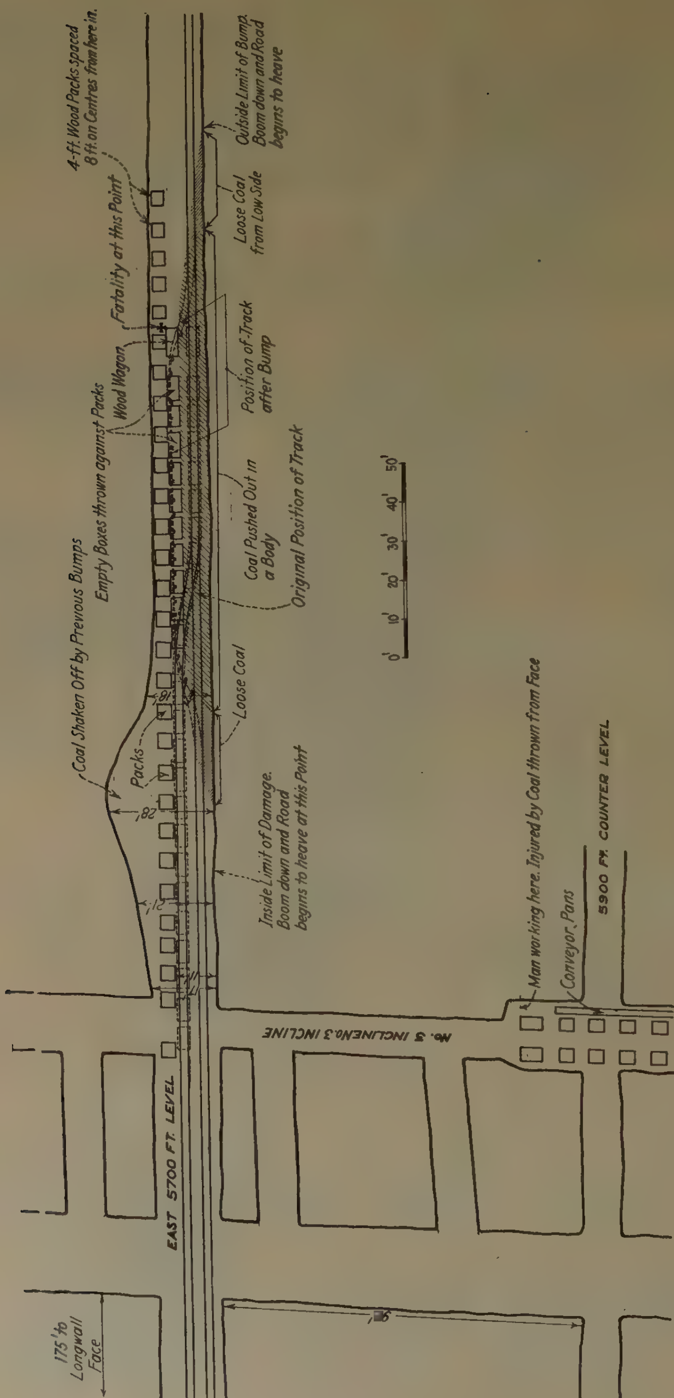


FIG. 10.—PLAN SHOWING EFFECTS OF BUMP OF JULY 3, 1928.

accompanying this bump was sharper than previous ones, deranging the ventilation by damaging doors and wooden stoppings. Officials' flame safety lamps on the wall faces were extinguished and caps blown off, but there was no strata movement at the face, which stood intact. The levels were recently stone-dusted and this dust went up the main slope in what was described as a white cloud. Considerable gas was given off

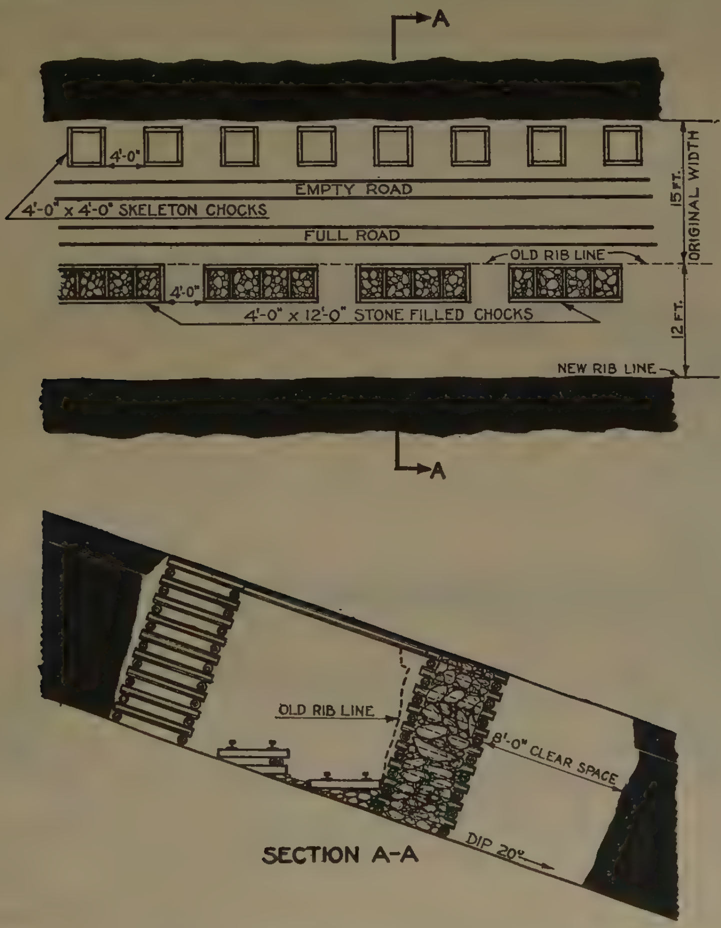


FIG. 11.—PLAN SHOWING WIDENED LEVELS.

and it was difficult to see more than a yard ahead in the roadways for some time afterwards on account of coal dust in the air. Like the previous district bump, it was felt distinctly on the surface.

Widening Levels

Experience with these two district bumps indicated that work at the wall faces was comparatively safe, the most dangerous zone being on the

main levels within 500 ft. of the working face, due to the sudden extrusion of coal from low side pillar and the breaking of the floor about the center of the roadway. To combat this danger it was decided to take 12 ft. off the low side rib and build substantial stone-filled packs next to the track, leaving a clear space of 8 ft. between new low rib and low side of pack into which the coal could extrude and thus give a chance of escape to anyone on the level, as shown in Fig. 11; this ribbing to extend from the face 800 ft. outbye. The possibility of bringing on other troubles by this widening was considered but the excellent roof condition suggested a trial.

October, 1928, District Bump

On Oct. 12, the wall faces having advanced 240 ft. since July, a third district bump occurred on the 5700-ft. level, with all the distinctive

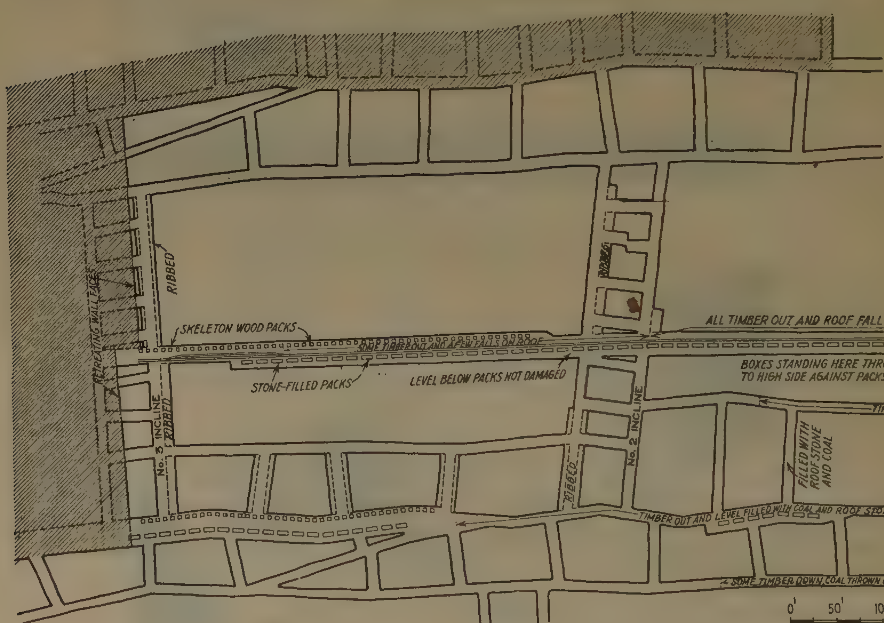
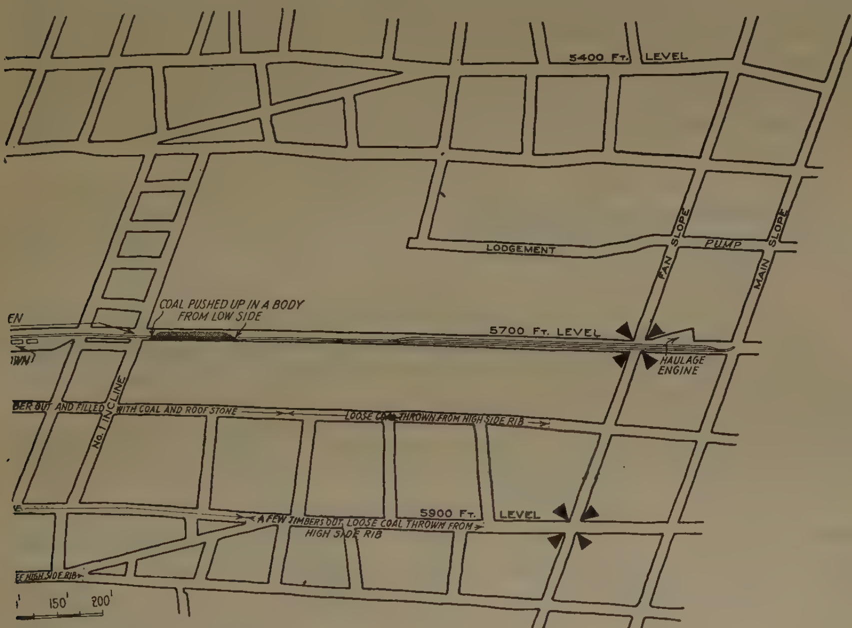


FIG. 12.—PLAN SHOWING EFFECTS

features of the previous two. In the interim one minor bump had occurred on the 5700-ft. level six days after the July district bump before the roadway was cleared and work recommenced, and three low side bumps of the first-mentioned order took place near incline 1 on the 5900-ft. level on July 24, Sept. 26, and Oct. 9, respectively. It may be noted that these three bumps and the district October bump all occurred in an area practically in line with the wall faces on the 5400-ft. east level which had been stopped in January, 1927 (Fig. 3).

A feature of the October district bump was the still greater distance from the wall faces at which the main exhibition of force appeared, in this case about 900 ft. The first sign of disturbance was the knocking down of props 150 ft. from the faces followed by damage of various kinds on the level for 1200 ft. further out. Coal was shaken off the high side of the 6300 and 6500-ft. levels and in all an area 1200 by 800 ft. was affected. It was noticed that short portions of the 5900-ft. level where the floor had shown a tendency to heave were unaffected by this bump.

Fig. 12 shows the position of the wall faces and records the condition of the roadways immediately afterwards. It will be noted that the low side ribbing previously referred to had been completed on the 5700-ft. level for a distance of 750 ft. and on the 5900-ft. level for 300 ft. at face and 120 ft. some distance further out. While this widening of the low side was the means of preventing fatalities or serious accident to men in



OF BUMP OF OCT. 12, 1928.

the area, it had not been extended sufficiently far to include the section of the roadway into which a portion of the low side pillar had been extruded, but as quantities of coal and roof stone for the first time fell from the high side on a large scale, it was further demonstrated that a similar widening on the rise side was desirable in future layouts with larger pillars, the traveling or haulage way running between two rows of stone-filled packs, with a space between each rib and side of pack.

It must be appreciated that up to the present time the areas mined by retreating straight faces had previously been developed for room and pillar work. The new blocking out for retreating is shown in Fig. 3, below the 6500-ft. level, the pillars measuring 230 ft. on the line of dip.

As the effects of this bump extended to within 60 ft. of the fanway, it appeared certain that further mining in this section would result in another district bump some three months later, which would wreck the main slopes and probably destroy the lodgment and pumping station, so, after clearing up the roadways the district was abandoned and longwall faces established at the boundary of the 6300 and 6500-ft. levels.

Classification of Bumps and Accidents, 1924–1928

Appendix 1 is a list of 117 bumps, recorded between Oct. 8, 1924, and Oct. 17, 1928. These bumps are successively numbered in order of occurrence and each has a corresponding number on Fig. 3, showing its location, the arrow indicating the direction of force. Several dates are on the plan to give an idea of the relative position of the bump location to the working faces during the 4-year period under review.

Table 1 shows by years the number and type of bumps, and accidents due to this cause.

Table 1.—*Classification of Bumps at No. 2 Mine, Springhill, N. S.*

Date	Months	High Side	Low Side	Longwall Face	Total	East Side	West Side	Fatalities	Injuries	Total
October, 1924, to December, 1924	3	8	6		14	5	9	2	1	3
1925	7 5 months' strike	12	11	1	24	15	9		6	6
1926	12	25	7	5	37	15	22*	1	15	16
1927	12	16		7	23	21	2		1	1
January, 1928, to October, 1928	10	10	9		19	19		1	9	10
Total	44	71	33	13	117	75	42	4	32	36

* West side stopped November, 1926.

The immunity from accidents in 1927 can be attributed to the west side and 5400-ft. east side workings being stopped, the east 5700 and 5900-ft. longwall at this period not having proceeded sufficiently far west to reach the district overlain and underlain by massive sandstone. During this year every bump came from the high side, indicating a readjustment of upper strata in areas previously mined in bumpy ground.

The bump accidents recorded occurred under the circumstances shown in Table 2.

TABLE 2.—*Accidents Caused by Bumps at No. 2 Mine*

	Fatalities	Injuries	Tota
Driving rooms or inclines	2	12	14
On longwall faces	1	7	8
In transit or working on main haulage levels	1	13	14
Total	4	32	36

The four fatalities occurred in four separate bumps and 32 men were injured in 18 bumps, the fatalities and injuries combined in 19 bumps. Although each bump was a potential accident, the actual ratio over 4 years was 6 to 1. Since July, 1926, no accidents from bumps have occurred on the longwall faces.

CAUSE OF BUMPS

These sudden burstings have occurred in several coal fields and are, under certain conditions, akin to the rock bursts in deep metal mining, which are ascribed to the release of internal strain in the rock masses caused by past compression or bending, or, in the case of the larger movements, to the overweighting of pillars on the edge of excavated areas causing a sudden collapse of the pillar or roadway bordering it. Bumps in coal mines may arise from several causes, of which released tectonic stress following extraction could be one and this would appear to be the only case in which depth would not be a material factor. Other causes, such as strata folding, intrusion of igneous rocks, high inclination, slippage on, or excessive local weight due to faults, pent-up gases, or mining under mountainous country causing unequal rock strain are conditions which do not apply at Springhill. The seam is regular, practically free from faults, is mined under level ground, and no gas outbursts have taken place. Possibly past methods of mining may be a contributory cause, but this is very doubtful. In driving development levels such as the 7400-ft. in the deepest portion of the mine and far removed from waste there is no tendency for pieces of coal to fly from the face and apparently the coal is normally strong enough to withstand the superincumbent burden and "works" freely.

The writer believes the bumps in seam 2 are the result of mining under and above a particular strata condition at considerable depth—that is, a very strong and tough immediate shale roof and relatively slightly weaker shale floor, which are respectively underlain and overlain

by massive sandstone beds as shown in Fig. 2—and that geogonetic tension is only of minor importance. Under these conditions bumps have frequently occurred with the room and pillar system of mining where the pillars have been extracted and under the longwall retreating method, where the waste has been supported on substantial pack walls placed at regular intervals. That this contention is correct seems substantiated by the fact that under like conditions of depth and mining systems, past and present, bumps do not occur where overhead drilling has shown the discontinuance of the sandstone bed or where the floor is weak enough to heave in the roadways. It may be coincidence but the overhead sandstone disappearance is generally accompanied by a weaker floor.

At depth under the strong roof and floor conditions previously described the writer is of the opinion that inclination of strata is not a material cause, but that for a given depth it intensifies the shock due to added local weight. Bumps of a similar nature would occur if the seam were flat.

The district or major bumps which have been described and illustrated and the bumps of lesser intensity which occurred in advance of longwall faces before any great area was extracted or when just entering a region overlain by massive sandstone are in the writer's opinion caused by a sudden reversal of strain (depth always being a factor) in the strata overlying and ahead of the coal face. As a wall face advances the immediate roof falls fairly well, observed falls of 8 ft or 10 ft. above the packs being regular and probably to the base of the overlying sandstone. This sandstone, being very strong and compact, has not the ability to fall or shear until a very considerable area has been excavated, during which time it is hanging back in the waste supported on a fulcrum at the coal face, resulting in an upward stress in this band and those above, reaching far over the solid coal. When this band finally breaks somewhere back in the waste, or possibly only slips very slightly on a fracture, the cantilever effect ceases, the rocks in upward strain over the coal are subject to a sudden reversal of stress and the coal ahead of the face is struck a sudden and hammerlike blow. This blow, due to the weight of overhead strata at considerable depth, exercises great force and although it may lower the roof only a fraction of an inch, the impact is such as to burst the solid coal into the roadways, break the floor, split pillars, and occasionally, owing to the jar, bring down roof stone. This force would appear to center at one point from which wave motions are sent out in all directions, these ceasing when no further resistance is encountered. Although the roof at the working face does not break there is generally evidence of extra pressure and a grinding action caused by the changing of the fulcrum point as the face advances. This shows as slight roof cracks, 4 or 5 ft. back from and parallel to the face, from which falls a small steady stream of rock dust as fine as flour.

Referring to the July and October district bumps under the conditions described, it would appear that the overlying sandstone breaks off in the waste after an advance of some 200 ft., about a three-month interval. That the advance was some 40 ft. greater in the October bump may be accounted for by increase in thickness of the sandstone in the direction of advance. This may also explain the greater distance ahead that the main exhibition of force was manifested in each successive major bump or that may be caused by the breaking of strata higher up. It seems, however, more reasonable to assume the blow was struck ahead in each case a distance proportional to the length of overhang prior to fracture, and if the ground at this point had already been loosened by a previous bump, the shock was transmitted to firmer ground where it found resistance. This may explain damage up to 1350 ft. ahead of the wall faces in the October district bump. In the case of a face being stopped a bump is liable to occur if and when the overlying strata break in the waste, as witness bump 260 on the 5900-ft. west level (Fig. 3). This bump occurred nine months after mining ceased on this side and struck a typical distance ahead of the wall faces.

It will be noted that the three district bumps of April, July and October exhibited their greatest force in the same straight line and relatively the same distance from the waste to the rise, possibly due to the right-angle line of waste forming a double overthrown arch. When the reversal of strain takes place there may be a definite point at the intersecting of the arches which is subject to the greatest reaction. It is believed the district bump of December, 1924, was caused by identical reversals of strain, to some extent intensified by the line of waste being almost parallel to the strike, making it harder for the overhead sandstone to break off in the waste because it was buttressed below. Here the waste contained no supporting pack walls. Generally it can be said that the introduction of waste packs will retard to some extent the breaking of the overhanging strata but without them, or with packs of insufficient strength, breaks at the face and frequent falls would result and also experience has shown that the working face is liable to bump.

It is quite possible that the bumps experienced in driving rooms almost skirting waste are caused by a gradual overloading of the intervening or adjacent pillar, this gradual overloading causing bending stresses in the roof strata to a point beyond which a sudden redistribution of forces takes place, this in turn causing a movement sufficient to produce a bump. On the other hand, many bumps in rooms in proximity to waste may be explained by the falling of the upper strata in the waste producing a reversal of stress over the coal in a similar manner but to a lesser degree than in a district bump—the driving of the room in such an area being coincident with the waste fracture. A waste bump which does no damage in the mine is probably the result of the breaking of hard rocks in

the upper strata, some considerable distance back in the waste, causing a small forward slippage of a portion of strata such as might occur when a long arch span broke. Some waste bumps may cause visible bumps, especially those from the high side some distance from totally extracted areas.

Subsidence

Bench marks down to hard rock were established over the workings and worked-out areas 4 years ago and levels taken every 6 months. Although during this period some 40 waste bumps, which caused no damage underground, were felt in the town situated beyond the seam outcrop, in addition to the recorded bumps, the bench marks have not been lowered a fraction of an inch. This is suggestive of very long arch spans in the upper rocks which are of strong hard texture.

Observed Characteristics of Bumps at Springhill

1. All bumps occur suddenly, make a loud report, and are without warning, except in rooms driving in the near vicinity of waste; the miners say that just prior to a bump the place becomes quiet. This knowledge has saved several accidents, the men retiring outbye at once. This applied more particularly to low side bumps. It is indicative of a change in stress as the normal weighting is taken care of when the coal is heard to "work."

2. A low side bump is of much greater intensity than one in which the force is from the high side. In the former case the coal offers greater resistance to the blow of strata readjustments which reacts through the floor and throws anything resting thereon to the high side. Timbers are rarely broken and seem to have been sprung out. Victims of low side bumps show typical shock symptoms.

3. The roof is seldom broken by a low side bump unless it is of major extent or in the case of a district bump in which a pillar is split.

4. A high side bump generally only throws coal from that rib and often breaks the timbers, there being no observed floor movement. A heavy high side bump will usually bring down roof and occurs some distance back from the face.

5. An air blast accompanies all district or major bumps, sometimes deranging the ventilation, and to a lesser degree air blasts occasionally occur with the minor low side bumps.

6. Gas may or may not accompany a minor bump but a considerable volume of gas is given off with a waste bump and with a district bump, which suddenly dislodges several hundred tons of coal. A few men have been gassed but probably only one fatality occurred through this cause and that happened when driving an incline in solid coal. The breaking of compressed air mains is a factor in diluting gas.

7. Severe bumps only occur in the vicinity of totally extracted areas.

8. A bump never reoccurs on a section of roadway which has previously been disturbed by a district or a low side bump, but occasionally a high side bump happens several times at the same point and a section of roadway in a district bump area is free from disturbance from the low side if the pavement has shown signs of heaving.

9. A major or waste bump is felt distinctly on the surface 2 or 3 miles from its point of origin.

10. After a district bump, it takes about 24 hr. for the affected area to quiet down. During this period the upper roof works heavily although a fall rarely occurs and much noise is heard in the upper strata, likened to the tearing of rocks apart. At the same time the pavement is subject to a series of oscillations, very distinctly felt when sitting on a prop on the floor, the feeling being as if one were jerked upwards. This would denote a series of strata adjustments similar but of lesser degree than the original bump, each one of which might cause a bump in firm strata; as, however, the ground has already been shaken and to some extent opened up, there is no resistance encountered by the waves from successive adjustments, consequently the movement is just discernable.

Suggestions to Mitigate the Effects of Bumps

As indicated, bumps may be of several types and result from different causes. With present knowledge it would be unwise to generalize and each case must be considered on the conditions and circumstances recorded. What may tend to a mitigation in one instance may be futile in another, and the economic factor is always present. Experience with bumps for a considerable number of years leads the writer to the opinion that under certain conditions of overlying and underlying strata no known method of mining will eliminate them and allow of coal production on a commercial basis, but it is often possible to establish conditions which may nullify or at least mitigate their effects. The following suggestions are based on the assumption that the theory advanced for their cause is correct.

1. If an upper seam exists within a reasonable distance of a lower seam which has roof and floor conditions likely to cause bumps, remove the upper seam first by the longwall method if it is economically feasible to do so, care being taken to leave no unworked portions in the upper seam. This will introduce a buffer to absorb the subsidence of the upper strata and cushion the effect of weight over a strong band of strata. Experience on the Rand is to the effect that "bursts do not occur in the workings of one reef if another above or below has been removed except where the parting is exceptionally thick."[1]

[1] Report of the Witwatersrand Rock Burst Committee, 1925.

2. Under similar conditions it would appear less advisable to mine an underlying seam because of the possibility of forming irregular spaces between either the overlying coal seam and sandstone beds or the strata immediately in contact with the seam and such beds. Irregular subsidence might here be a factor which would cause greater movement in the strong strata overlying the upper seam when it came to be mined. It is, however, under certain conditions a suggested remedy worthy of consideration.

3. In the event of neither of the foregoing suggestions being feasible or accomplishing the desired result, the necessity for mining advancing longwall or retreating longwall has a bearing on the type of remedy to adopt. Assuming the former is adopted the resilience of the waste and the roadways constructed therein would be such as to practically preclude the possibility of bumps in that zone destroying roadways On the other hand, a sudden reversal of strain in the immediate strata over the solid coal some distance ahead of the working face might result in the force extending to the face throwing off coal and causing floor upheaval, as previously described and illustrated in a roadway in solid coal, but possibly with more disastrous results.

It is stated in one instance where serious face bumps have occurred with advancing longwall that generally these happened when the face was parallel to the dip and the suggestion was made to keep faces parallel to the strike and advance uphill as the roof should fracture more readily and result in less weight thrown forward.[2] This system must presuppose considerable dip development to make the lifts as long as possible but even under the best conditions a point is reached before going through on waste to the rise where an intervening slice of coal will introduce grave difficulties. It might be advisable to abandon this coal but should such a course lead to other troubles it is suggested it should only be mined by shooting, at which time a minimum number of men would be in the district. If a sandstone band immediately overlies the seam, waste fracture would be helped by shooting the roof periodically. Possibly under certain conditions it might be advisable to shoot both roof and coal as the normal method of mining. Tight waste packing is essential to cushion the waste falls and make possible roof control at the working face.

4. On the supposition that longwall retreating is the only economically feasible mode of work—as with the No. 2 seam, Springhill—because of the thickness of seam and lack of stowing material, the writer does not believe it practical to retreat uphill with faces parallel to the strike, owing to the difficulty of residual pillars at the top end, a condition which should be avoided at all cost, and retreating downhill would throw extra weight on the roadways. From the point of view of extraction,

---

[2] Report issued in 1927 by British Mines Department on upheaval of floor at the Pendleton Colliery, Lancashire.

safety and transport, the system now in use with faces about 230 ft. long, parallel to the dip, appears the most suitable and it has been practically demonstrated that with substantial waste packwalls and open roadways ahead of the faces face bumps do not occur and the working face is a comparatively safe zone.

To counteract the effects of bumps in main roadways leading from face, it is suggested in the first instance to widen the levels and introduce stone-filled packs both on low and high side of roadway as previously described. Should this prove ineffective it is then suggested to loosen the low side rib by blasting. Holes, say 20 ft. long, would be drilled 15 ft. apart in the center of the seam and fired with heavy charges, the rib to be thus loosened for at least 1000 ft. from the wall faces. It is possible this blasting may induce bumps but the shotfirers would be stationed at a place of safety. It is the writer's opinion that it is possible to have the pillars too strong ahead of a retreating wall face, a condition that would bring about face bumps. To avoid this shooting the low side rib may be necessary.

Probably the best solution and one which would make for comparative safety in mining a seam liable to bumps by the longwall retreating system, provided the seam is of sufficient value, is to drive the main roadways in the strata under the seam, leaving about 20 ft. between the floor of seam and roof of main road, immediately over which and for 20 ft. on either side of the center line of main road the overlying seam would be extracted and two rows of stone-filled packs placed therein, a method adopted in some deep metal mines.[3] The effect of bumps would then present no transportation difficulty as the coal from wall faces could be taken out a distance by belt conveyor to a chute connecting to main road in understrata.

5. The writer would stress the advisability of lodgments and pump rooms being constructed in the strata under the seam and extracting the coal immediately overhead and some little distance beyond the sides of the roadway below, placing stone-filled packs in the seam excavation. The need for this was demonstrated when the October district bump opened up the ground disquietingly near the lodgment.

Hydraulic Stowing

Sand flushing in certain circumstances might be a complete remedy, but unless circumstances are exceptional it is outside the bounds of practicability. Material of a less compact nature than sand would allow of sufficient settlement to fracture a very strong roof and there is doubt as to the complete efficacy of sand, especially if the seam is of slight inclination.

[3] Report of the Witwatersrand Rock Burst Committee, 1925.

Drilling

To determine what areas are likely to give trouble from bumps and where it may be necessary to take the precautions outlined, it is recommended that drill holes at regular intervals be put up and down for a distance at least 100 ft. in each case to test the nature of the overlying and underlying strata.

Acknowledgments

The writer wishes to acknowledge his indebtedness to T. L. McCall, Assistant Mining Engineer, A. K. McLeod, Resident Superintendent, and Hugh Peppard, Chief Surveyor of the Cumberland Railway Coal Co., for detailed information supplied and help in the preparation of drawings.

Appendix 1.—*Bumps in No. 2 Mine, Springhill, Nova Scotia, Oct. 8, 1924, to Oct. 17, 1928*

No.	Date	Location	Remarks
176	Oct. 8, 1924	5400-ft. west level, incline 4, room 2 east.	Force from high side; 3 sets timber and some stone down.
177	Oct. 9, 1924	5400-ft. east level, incline 4, room 1 east.	Force from low side; coal thrown out.
178	Dec. 1, 1924	5400-ft. west level, top of incline 1½.	Force from high side; a number of rib props out. Considerable coal thrown out.
179	Dec. 2, 1924	5400-ft. west level, incline 2, room 2 west, at face and high side.	Force from high side, coal thrown from high side and face.
180	Dec. 2, 1924	5400-ft. west level, full end turnout 2.	Force from high side, some high side props, 3 booms, some coal and stone down.
181	Dec. 2, 1924	5400-ft. west level, incline 1½, on west rib about 160 ft. below face.	Force from high side; a number of rib props broken and coal thrown out.
182	Dec. 2, 1924	5400-ft. east level, incline 4, room east at face.	Force from low side; 3 sets timber, 4 low side props, 3 boxes* of stone and some coal down.
183	Dec. 5, 1924	5400-ft. east level, incline 6, pillar 1 east, at face of cut.	Force from high side; knocked out running boom and 2 sets up the cut. Ten boxes of stone down.
184	Dec. 6, 1924	5400-ft. west level, incline 1, above and below 4700-ft. low level.	Force from low side; a large number of east side rib props out and broken. Large amount of coal off ribs, also fall of stone in low level.
185	Dec. 6, 1924	5400-ft. east level, between inclines 3 and 4, also inclines 3 and 4 and top rooms.	Force from low side; level and incline wrecked; timber thrown out, roof and floor broken. (District bump.) Felt distinctly on surface; one fatality.
186	Dec. 8, 1924	5400-ft. east level, from door between inclines 2 and 3 into turnout 3.	Force from high side; timber thrown out and road damaged for 50 or 60 ft. Packs moved down hill.
187	Dec. 12, 1924	5900-ft. west level, incline 6, top rooms.	Force from low side; coal and track thrown to high side of room; one fatality and one man injured.
188	Dec. 29, 1924	5900-ft. west level at incline 4.	Force from low side; wrecked 100 ft. of road, knocked out 6 sets timber and a number of boxes of coal and stone down; felt on surface.
189	Dec. 29, 1924	5900-ft. west level on level between inclines 6 and 6½.	Force from high side; few tons of coal knocked out.
190	Jan. 3, 1925	5400-ft. east level, on level at slant 2.	Force from low side; level full of coal for about 40 feet.
191	Jan. 6, 1925	5400-ft. west level, level and counter level at incline 2½.	Force from low side; level full of coal and roof stone down, counter level damaged. Two men injured.
192	Jan. 8, 1925	5400-ft. east level on level 100 ft. outside incline 3.	Force from high side; 8 high side props and 30 boxes of coal down. Packs and high side track damaged.
193	Jan. 9, 1925	5900-ft. west level, incline 4, east side of incline 100 ft. above counter level.	Force from low side; 12 sets of timber knocked out, some coal and stone down.
194	Jan. 9, 1925	5900-ft. east level, on level 200 ft. outside incline 6.	Force from high side; 12 high side props broken, 8 tons of coal down.
195	Jan. 13, 1925	5400-ft. east level, head off level inside incline 3.	Force from high side; coal off west rib and face, 8 west side props knocked out at face; one man injured.

* Mine box holds approximately 2000 pounds.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
196	Jan. 14, 1925	5400-ft. east level, counter level 100 ft. east, incline 3.	Force from high side; a large amount of coal off high side rib for 75 ft.; no stone down.
197	Jan. 16, 1925	5400-ft. east level, counter level east incline 3 at face and along low side for about 50 ft.	Force from low side; 3 sets of timber and some stone down. Road thrown up for 50 ft.; heavy bump.
198	Jan. 16, 1925	5400-ft. east level head off level	Force from high side; did considerable damage to counter level.
199	Jan. 28, 1925	5900-ft. east level, slope pillar, bottom turnout inside end.	Force from high side; knocked out 6 high side props and 5 or 6 boxes of coal.
200	Feb. 2, 1925	5400-ft. west level, incline 3	Force from low side; incline in solid (very heavy shock on surface).
		Strike, March 5 to Aug. 6, 1925.	
202	Mar. 13, 1925	5400-ft. east level, ribbing first head down inside incline 2.	Force from high side; coal off ribs, top coal and stone down, timber knocked out.
203	Aug. 21, 1925	5400-ft. west level, incline 2½, face top room east.	Force from low side; some coal off low side and road thrown up.
204	Aug. 27, 1925	5900-ft. west level, incline 5½ face.	Force from high side; knocked out 8 sets of timber, 8 tons rock down at working face.
205	Aug. 29, 1925	5400-ft. west level, incline 2½, top room east; in high side from face to cope up.	Force from high side; coal thrown out from face to landing.
206	Sept. 2, 1925	5900-ft. west level, incline face 5½; incline up 300 feet.	Force from high side; knocked out wheel prop and injured man running wheel.
2207	Sept. 4, 1925	5400-ft. east level, longwall from level to counter level.	Force on longwall face; large amount of coal off face, conveyor pans thrown over against packs. Two men injured.
208	Sept. 11, 1925	5400-ft. east level, first head inside No. 3; incline west rib.	Force from high side; about 20 boxes coal off west rib.
209	Oct. 19, 1925	5400-ft. east level, incline 2½, 30 ft. from face.	Force from low side; set of timber down, road thrown up.
210	Oct. 19, 1925	5400-ft. west level, incline 2½, top room west.	Force from low side; 30 ft. of road thrown to high side.
211	Oct. 26, 1925	5900-ft. east level, incline 10, room 1 east, 25 ft. from face out.	Force from low side; knocked out 6 low side props and 2 sets of timber, damaged 25 ft. of road.
212	Oct. 30, 1925	5900-ft. west level, incline 5½, half-way level west at face.	Force from high side; no damage. Coal thrown from face and high side rib.
213	Nov. 27, 1925	5400-ft. east level at incline 2½ turn-out.	Force from low side; 11 low side props knocked out and 75 ft. of road damaged.
214	Jan. 5, 1926	5400-ft. west level, inclines 2 to 2½, room 1.	Force from high side; packs thrown to low side and roof fallen for 150 feet.
215	Jan. 11, 1926	6500-ft. east level, on level 50 ft. from main slope.	Force from high side; 4 props and 6 boxes of coal knocked out.
216	Jan. 16, 1926	5400-ft. east level, head off level	Force from high side; did considerable damage to counter level. Two men injured
217	Feb. 1, 1926	5400-ft. west level, room between inclines 2 and 2½, about 25 ft. from face, room ribbed in for 90 feet.	Force from high side; 10 props knocked out and packs damaged. Large stone slid down on road; two men injured.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
218	Feb. 23, 1926	5400-ft. east level, rib up old head outside longwall, along west rib bottom to face, head up 60 feet.	Force from high side; all west rib props and booms down, 15 boxes of coal knocked out.
219	Feb. 27, 1926	5400-ft. west level, incline 2½, between level and counter level.	Force from high side; timber and coal thrown out.
220	Mar. 2, 1926	5400-ft. west level, low side of level outside incline 2.	Force from low side; 4 low side props knocked out.
220-A	Mar. 13, 1926	5400-ft. east on level between longwall and incline 2.	Force from low side; level closed at middle of bull wheel turnout.
221	Mar. 30, 1926	5400-ft. west level, incline 2½, from counter level up 100 feet.	Force from low side; closed for 40 ft., props broken for 60 feet.
222	Mar. 30, 1926	5900-ft. west level, ribbing in low level from incline 3.	Force from high side; old low level brought in, some stone fell on high side of level for 100 feet.
223	April 1, 1926	5400-ft.west level, incline 2, top room west, 60 ft. from incline.	Force from high side; brought in roof stone for 100 feet.
224	May 4, 1926	5900-ft west level, incline 3, halfway level along high side from face out 30 ft., level in 360 ft. from incline.	Force from high side; knocked out 9 high side props, 6 center props and about 10 tons of coal. One man injured.
225	May 18, 1926	5400lft. west level, longwall, 100 ft. above counter level ribbing incline 2½.	Force on wall face; bumped coal off wall.
226	May 19, 1926	5900-ft. west level, incline 3, halfway level at face.	Force from high side; knocked out 1 prop and about 10 tons of coal. One man injured.
227	May 19, 1926	5900-ft. west level, incline 3, halfway level from face out 50 ft. high side.	Force from high side; knocked out 8 high side props, 50 tons of coal and 4 tons of stone down.
228	June 5, 1926	5400-ft. west level, ribbing incline 2½ from 40 to 70 ft. above counter level.	Force from high side; coal knocked off rib for 30 feet.
229	June 7, 1926	6500-ft. east level, opposite incline 6 on level.	Force from high side; 13 sets of timber down, some stone and coal from high side rib.
230	June 14, 1926	5400-ft. west level, longwall 110 ft. above counter level.	Force on longwall face; threw out 2 packs, 12 boxes of coal. Two men injured.
231	June 16, 1926	5400-ft. west longwall at face of ribbing up incline 2½, 120 ft. above counter level.	Force on longwall face; no damage. Face filled with coal. One man injured.
232	July 2, 1926	5400-ft. west level, longwall face 120 ft. from level to top.	Force on longwall face; shifted 5 hardwood packs, knocked considerable coal off face. One fatality and two men injured.
233	July 5, 1926	5400-ft. west level, room from inclines 2 to 2½.	Force from high side; packs thrown to low side, room fell in for 150 feet.
234	July 6, 1926	5400-ft. west level, on level turnout at longwall.	Force from low side; 4 sets of timber knocked out, top coal and stone down, road thrown up.
235	July 11, 1926	6500-ft. east level, main level between fanway and main slope.	Force from high side; 4 props out and 6 boxes of coal down.
236	July 13, 1926	5400-ft. west level, 75 ft. inside incline 2 on high side.	Force from high side; 2 sets timber, 2 boxes stone and 3 boxes coal down.
237	July 16, 1926	5400-ft. west level, on level between inclines 1½ and 2.	Force from high side; 4 sets of timber down, 20 high side props broken, 13 boxes of coal thrown out.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
238	July 17, 1926	5400-ft. east level on level between inclines 1 and 2.	Force from high side; 2 sets of timber down, 15 high side props broken, 9 boxes of coal from high side.
239	July 19, 1926	6500-ft. east level between fanway and main slope.	Force from high side; knocked out 3 sets timber, 2 high side props; 8 boxes of coal thrown out.
240	Aug. 20, 1926	5400-ft. west level, incline 2, from counter level up.	Force from low side; road thrown up and props broken.
241	Oct. 4, 1926	5400-ft. east level on level 60 ft. inside incline 1.	Force from high side; threw two trams of pack timber from high to low side.
242	Oct. 11, 1926	5900-ft. west level, on level at incline 2 turnout.	Force from low side; 4 booms down, several low side props out and 10 boxes of coal off rib.
243	Oct. 29, 1926	5400-ft. east level, on level 25 ft. outside longwall.	Force from low side; 80 ft. of level in, coal and stone down; first bump outside wall faces on east side.
244	Nov. 6, 1926	5400-ft. west level, outside bull wheel turnout for 50 ft. out.	Force from high side; 2 sets of timber and 2 boxes of coal knocked out.
245	Nov. 25, 1926	5400-ft. east level, on level outside end of bull wheel turnout.	Force from high side; 5 sets of timber and 18 boxes of coal knocked out.
246	Nov. 26, 1926	5400-ft. east level, on level 200 ft. outside longwall.	Force from high side; 8 boxes of coal, some high side props and 2 booms knocked out.
247	Dec. 1, 1926	5400-ft. east level, on counter level 60 ft. from longwall face.	Force from high side; 3 sets timber down, coal off high side, top coal and some stone knocked out.
248	Dec. 3, 1926	5400-ft. east level, bull wheel turnout, 30 ft. out from longwall face.	Force from high side; number of props broken and some coal down.
249	Dec. 11, 1926	5400-ft. east level, longwall face from level to counter level.	Force on longwall face; knocked out 20 tons of coal; probably a face weighting.
250	Dec. 19, 1926	5400-ft. east level, 30 ft. outside longwall face.	Force from low side; closed 2 air heads off low level and broke door in slant. Bumped low level pillar, brought in 60 ft. of roof stone in main level and threw 4 packs to low side of low level. This bump very noticeable on surface. Four men slightly injured.
251	Feb. 28, 1927	5900-ft. east level, longwall face from counter level to 5700-ft. level.	Force on longwall face; 40 ft. of face damaged, stone down and props broken, probably a face weighting.
252	Mar. 3, 1927	5700-ft. east level, about center of longwall face.	Force on longwall face; broke several booms and props, threw coal off face, probably a face weighting.
253	Mar. 21, 1927	5700-ft. east level, longwall face.	Force on longwall face; coal knocked off face and 3 props broken; probably a face weighting.
254	Mar. 25, 1927	5900-ft. east level, inside of bottom turnout.	Force from high side; two low side props and some coal down on high side.
255	Mar. 25, 1927	6500-ft. west level, on main level between main slope and pipe slope.	Force from high side; 5 boxes of coal knocked down from high side rib.
256	Mar. 25, 1927	5700-ft. east level, longwall face.	Force on longwall face; knocked down coal and broke roof along face; probably a face weighting.
257	Mar. 30, 1927	5700-ft. east level, longwall face.	Force on longwall face; coal knocked off face of wall; probably a face weighting.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
258	Mar. 31, 1927	5700-ft. east level, bull wheel turnout.	Force from high side; 3 sets of timber out and 2 boxes of stone down, broke air line.
259	April 23, 1927	5900 and 5700-ft. east levels.	Force from high side; number of booms broken on bull wheel turnout.
260	Aug. 12, 1927	5900-ft. west level, incline 4½.	Force from high side; 8 high side props broken, 10 boxes of coal off high side rib, large fall of stone on incline 4½.
261	Sept. 16, 1927	5700-ft. east level, on bottom turnout near fanway.	Force from high side; 4 boxes of coal off high side rib.
262	Oct. 14, 1927	5700-ft. east level, along longwall face.	Force on longwall face; broke roof along face of wall; probably a face weighting.
263	Oct. 27, 1927, 1:30 p.m.	6500-ft. east level, incline 8, room west.	Force from high side; coal knocked down. One man injured.
264	Oct. 27, 1927, 2:30 p.m.	6500-ft. east level, on level near main slope.	Force from high side; coal knocked down.
265	Oct. 27, 1927, 2:45 p. m.	6500-ft. east level, on level outside incline 8.	Force from high side; coal knocked down.
266	Nov. 11, 1927	6500-ft. east level, at first head outside incline 1.	Force from high side; knocked out 16 high side props, broke 8 props and bumped about 35 boxes of coal off high side rib. Roof, pavement and low side rib not disturbed. Threw some coal off high side in counter level.
267	Nov. 18, 1927	6300-ft. east level, at incline 6.	Force from high side; 13 boxes coal off high side rib, 3 high side props and 8 booms broken.
268	Nov. 19, 1927	5900-ft. east level, longwall face 40 ft. from 5900-ft. level.	Force on longwall face; 8 boxes of coal knocked down from face in space of 15 ft.; no break in roof or pavement, no timber down.
269	Nov. 21, 1927	5900-ft. east level, 125 ft. outside longwall face between inclines 4 and 5.	Force from high side; 47 boxes of coal off high side rib, 30 high side props out, knocked empty boxes to low side, 50 ft. along high side damaged.
270	Dec. 15, 1927	6500-ft. east level, 150 ft., inside incline 8.	Force from high side; broke 5 booms, knocked out 2 boxes of coal off high side rib and 1 box of stone down.
271	Dec. 16, 1927	6300-ft. east level, 100 ft. inside incline 6.	Force from high side; 2 high side props out and about 8 boxes of coal knocked off high side.
272	Dec. 20, 1927	5900-ft. east level, on level 150 ft. inside incline 3.	Force from high side; knocked out 8 props and 10 boxes of coal.
273	Dec. 31, 1927	5900-ft. east level, 300 ft. outside longwall face. Face was 70 ft. outside incline 5.	Force from high side; 5 sets of timber knocked out and 8 boxes of coal off high side rib.
274	Jan. 6, 1928	5700-ft. east level, 50 ft. outside incline 3 back head.	Force from high side; moved 3 packs and knocked 25 boxes of coal off high side rib. One man injured.
275	Jan. 7, 1928	5900-ft. east level, 60 ft. outside incline 4, 250 ft. outside longwall face.	Force from high side; 18 high side props out, about 25 boxes of coal off high side rib. No stone or top coal down.
276	Jan. 7, 1928	5900-ft. east level, 70 to 80 ft. outside incline 4.	Force from high side; knocked out 30 booms and about 40 high side props, about 70 boxes of coal down. Two men injured.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
277	Jan. 23, 1928	5900-ft. east level, main slope and fan-way pillar from counter level to 5900-ft. level, and along 5900-ft. level from slope for 30 feet.	Force from high side; one boom and one prop broken on slope, 40 ft. above 5900-ft. level; one prop broken on level, 2 boxes of coal off rib.
278	Feb. 8, 1928	6300-ft. east level, 300 ft. outside incline 6.	Force from high side; 16 boxes of coal off high side rib, 3 high side props and 2 booms broken.
279	Feb. 29, 1928	5700-ft. east level, incline 3, from level for 50 ft. down incline.	Force from low side; 5 boxes of coal from low side 5700-ft. level, 50-ft. fall of stone in incline 3.
280	Mar. 24, 1928	5900-ft. east level, between incline 3 and first head outside.	Force from low side; bump was from high side on level and low side in counter level, bottom thrown up on level and counter level, 22 props and booms and 90 boxes of coal down, no roof stone down, 30 ft. of road on level thrown up; heavy bump.
281	Mar. 27, 1928	5900-ft. east level, on level from incline 3 in 100 feet.	Force from high side; 33 sets of timber knocked out and 100 boxes coal down.
282	Mar. 30, 1928	5700-ft. east level, pillar west side incline 3 from 5700 to 5400-ft. mine bord.	Force from low side; knocked all packs from west to east side of incline, some high side props and coal down on 5700-ft. level, 22 booms and some coal down on 5400-ft. low level at top of incline. This bump felt very distinctly on surface.
283	Mar. 30, 1928	5900-ft. east level on main level at second slant from slope.	Force from high side; several props and 12 boxes of coal knocked off high side.
284	April 12, 1928	5900-ft. east level, from near wall face out to incline 3 back head, a distance of 300 feet.	Force from low side; 300 ft. of level damaged, timber down and level filled with coal, road thrown to high side, portion of low side pillar shoved up hill 7 ft. District bump, felt distinctly on surface.
285	June 29, 1928	6500-ft. east level, 200 ft. outside slant 5.	Force from high side; 6 boxes of coal knocked off high side rib. No timber broken.
286	July 3, 1928	5700-ft. east level, from 30 to 220 ft. outside incline 3.	Force from low side; 190 ft. of level nearly closed. District bump; one fatality and two men injured; felt distinctly on surface.
287	July 9, 1928	5700-ft. east level, on level from incline 2 in 50 ft. to back head.	Force from high side; 8 boxes of coal off rib.
288	July 24, 1928	5900-ft. east level, on level from incline 1 to 50 ft. outside.	Force from low side; about 6 boxes of coal down.
289	Sept. 26, 1928	5900-ft east level at incline 1.	Force from low side; 8 low side props knocked out, road thrown up for 28 ft., 5 boxes of coal down.
290	Oct. 9, 1928	5900-ft. east level, 150 ft. inside incline 1.	Force from low side; 50 boxes of coal down, 17 sets of timber knocked out, air line broken, 100 ft. of track damaged and 18 ft. of low side track thrown to high side.

APPENDIX 1.—(*Continued*)

No.	Date	Location	Remarks
291	Oct. 12, 1928	5700-ft. east level	Force from low side; start of damage on level 150 ft. outside longwall face (incline three) extending outbye for a distance of 1350 ft. Maximum force exhibited at 900 ft. outbye longwall face. This bump wrecked portions of 5900-ft. level and threw coal from high side of 6300 and 6500-ft. levels. District bump; four men injured. Felt very distinctly on the surface.
292	Oct. 17, 1928	6500-ft. east level, 1450 ft. from slope.	Force from high side; 8 props broken, 12 boxes of coal down.

DISCUSSION

G. S. RICE, Washington, D. C.—This paper deals with an unusual mining difficulty, a natural condition which fortunately is found in few of the coal-mining districts.

As mentioned by Mr. Herd, I studied the phenomena in connection with bumps in September, 1924, in making a report jointly under the auspices of the mining company and the Minister of Public Works and Mines. This report indicated the necessity of changing the mining system to a longwall retreating method to lessen the danger of the occurrence of disastrous bumps.

Necessarily such a radical change of mining method applied in an old mine and from which it was necessary to keep on producing coal had to be taken by successive steps. Meantime the mining conditions under which "bumps" previously occurred still partly prevail—I refer to the presence of old pillars. It is therefore my feeling that the change of mining system as yet has not been sufficiently advanced to determine the success or failure of the proposed method of a complete longwall retreating system. At present it appears that the method is largely one of extracting previously formed pillars by "long faces" which are so separated and restricted in length that there is no opportunity for a complete fracture of the natural arches which tend to form in the strong rigid rocks overhead. The real test of the retreating longwall system will not come until there is a long line of connected faces with short steps between, of a length approaching the depth of coal from the surface. This comment is not in any way intended to reflect upon the splendid manner in which this difficult change of system of mining is being carried on.

I have not visited the Springhill mine since 1924 and therefore would like to ask a question about the method of making the pack walls. Where do you get the stone or rock for the pack walls behind the face?

T. L. MCCALL, Glace Bay, N. S.—We get that from the fallen rock in the waste. Wooden chocks are put in to steady the waste packings on the low side.

G. S. RICE.—Are those chocks filled?

T. L. MCCALL.—Yes, they are completely filled with stones.

H. G. MOULTON, New York, N. Y.—Is the theory underlying this that very strong hardwood chocks are put in to force the breaking of the roof back a given distance? That is, the strength of this line of hardwood chocks is such that the break would have to come behind them, and then there is sufficient compressible material to allow the failure to take place gradually?

T. L. McCall.—That is it. Sometimes the roof does not break readily. In that case, if you walk back about 70 or 80 ft. in the waste you will find the floor coming up and the roof coming down.

G. W. Evans, Seattle, Wash.—What is the relative nature of the roof and the floor? Is the floor softer than the roof?

T. L. McCall.—Yes. They are both hard, but the floor is slightly softer.

G. S. Rice.—How long before the bump of April 12 had the ground indicated in Fig. 6 been taken out?

T. L. McCall.—About 18 months or two years before. Our present knowledge indicates that we should have taken the coal out, because as far as our experience goes a district that has been bumped once will not bump again, but at the time we did not know that.

R. D. Hall, New York, N. Y.—What are the straight lines in the center of Fig. 6?

T. L. McCall.—Those are little faults; one has a displacement of 4 in., the other, 8 inches.

G. S. Rice.—Are they inclined or vertical faults?

T. L. McCall.—Nearly vertical.

G. S. Rice.—I might explain the steps which led to my recommending longwall retreating as a method of preventing disastrous bumps after visiting and studying the situation in the Springhill mine in 1924.

Various mining methods had been tried and pillar systems had failed. I was impressed with these outstanding things—I am speaking of the large features only—(1) There was a thick, strong bed of coal; (2) it had, at least in most places under consideration, a strong roof and fairly strong floor, both stronger in crushing strength than the coal; (3) the bed, being free from partings, did not provide material for pack walls in any plan of advancing longwall.

These were my observations in going around the pillar faces: Judging from the excellent way the roof was breaking where they were extracting the pillars, there was relatively little danger of the roof cutting off suddenly, close to the face. The roof in the pillar extraction was breaking diagonally upward across the bedding over the goaf and receiving support from the broken rock. On the other hand, they were having trouble in driving headings into large blocks of coal, due to the heavy pressure. Bumps were occurring there and some distance back from the pillars in the roadways. The mine was getting deeper all the time and as the depths increased the rock pressure increased, and so did the difficulty and danger of the bumps.

In going over the general mining experience of the past, it had been my observation in other districts, in America and in Europe, that bumps are associated with deep workings where the system of mining used coal pillars and where there was a strong roof, fairly strong floor and strong coal; where any one of these was weak, bumps were not probable, because the weaker material would give way gradually, and only squeeze effect would result.

For example, in Colorado and New Mexico, in mines with which I used to be connected, which worked under high mountains, there were strong roofs and strong floors in many cases, but with coal so jointed by slip planes that even when advancing entries into new territory, you would often see the coal ribs in thick coal slabbing off into the roadway under the pressure.

Again, it had been my observation that where a roof had fractured to the surface and had settled so it rested firmly on the broken mass of rock in the gob, bumps or rock bursts were unlikely. Thereafter you were dealing with a relatively loose overburden material if mining continued actively and systematically. Further, the only way such work can be safely accomplished is by longwall, either advancing or retreating.

Advancing longwall had to be discarded in this case because there was not available underground the necessary pack-wall material for a thick bed. Also, as Mr. McCall pointed out, they did not have a source of supply of sand or gravel for hydraulic stowing, a method that is so successfully applied in Germany and some districts in France.

The conditions were growing more serious with increasing depths. They had already a vertical depth of about 3000 ft. and there was 4000 or even 5000 ft. ahead of them.

By a process of elimination, retreating longwall was selected, and it was then a study as to how it should be applied. For the reasons that Mr. McCall has given, I dismissed in considering the questions retreating uphill or downhill, and therefore proposed retreating step faces parallel with the dip of the coal.

It was the mine owners' problem to put into effect this recommendation which they adopted in principle, and they had a most difficult one because they could undertake it only piecemeal. They had to deal with an old mine, and they had to maintain their production.

I believe that some of the bumps that occurred shortly after longwall retreating was started in isolated blocks of coal were incident to changing the method. I did not feel that it was a test of a completely developed system of retreat with connected longwall faces by which the arching of the strong overburden would be broken and thus relieve the pressure on the coal face. Apparently surface subsidence had not occurred, or at least so as to be noticeable. To obtain satisfactory longwall conditions the breaks or subsidence should extend up through the entire overburden to the surface, to prevent either cantilever or arching stresses.

The definitions of different types of bumps given in the paper are classed by violence and by size. I think it is preferable to classify by cause. As the result of my studies of bumps in various coal fields in the western part of this country, as in the State of Washington, but especially in reporting on the Crow's Nest Pass mines, British Columbia, in 1917 and on the Springhill mine in 1924, I group the different kinds of "bumps" into two general classes; *viz.*, "pressure bumps" and "shock bumps," the former being the result of overloading a pillar until it bursts, the other where a shock wave is transmitted through the rocks more or less at right angles to the bedding, like an earthquake wave, the shock being caused by the fall of a great mass of rock from some higher strata, through a space made by prior subsidence.

Some of the "bumps" at Springhill I considered "pressure bumps;" others, like that of Apr. 12, 1928, described in Mr. Herd's paper, I would call a "shock bump."

In the United States some bumps have occurred in the anthracite district (Pennsylvania) and in the far western coal mines but so far we have experienced few disastrous bumps in coal mines. In some of the deeper copper mines of Michigan there have been serious bumps; what are generally termed in metal mining, "rock bursts" or "air blasts." However, I fear that we are likely to experience more serious bumps in coal mining in the future if we continue to use pillar systems when the mines get under deeper cover, especially in the Rocky Mountain region and in the mountain districts of Washington where bumps have already occurred. I believe there is special danger, if suitable methods are not used, in deep mining of pitching beds whether the beds are coal or other mineral where the roof or hanging wall and the footwall are strong.

Some years ago, together with George Watkin Evans, I examined a place in the Carbonado mine, Washington, in which a bump had occurred, where the pillars of some rooms going up a steep pitch had crushed, nearly closing the openings, but the roof and floor remained intact. This I would class as a pressure bump. On another occasion we examined a place where a bad bump had occurred in the Carbonado mine, in a pitching bed with strong roof and floor. In this case there was distinctly a wave movement of the roof. Some room pillars had been crushed and in the level entry below there were two remarkable manifestations of wave motion; the roof was not broken yet a timber crib had been moved from the upper side of the entry to the lower side and was intact and tight against the roof, and at the low side rib, over the coal which had been crushed down 4 or 5 in., there was a space between it and the solid roof which extended in 3 or 4 ft. This I would attribute to the effect of a shock bump originating in a profound ground movement higher up due to subsidence where some pillars had been withdrawn in the neighborhood. This case was complicated because this part of the mine occupied a pitching anticlinal structure.

Mr. Evans will perhaps be willing to give us later information about bump conditions in these mines.

G. W. Evans.—We have had several different types of bumps in the Northwest. On Vancouver Island, at the Cassidy mine, we have sonething like bumps in effect, which are instantaneous outbursts of gas. At Black Diamond mine, we have bumps that are due almost entirely to roof pressure. Strong roof and a weaker floor, but strong coal under a cover of 2000 ft. We have had a number of fatalities at that mine. In one instance three men were killed and three locomotives were buried.

At Carbonado we have a simpler condition. Under about 2000 ft. of cover, the bed dips at an angle of about 35°, and there again we had a strong sandstone roof and a weaker floor, with strong coal in the Windgate seam. In that instance the movement was up the pitch.

I was at Crow's Nest Pass a couple of weeks ago, and one inspector, Jack McDonald, told me that they had had a bump a few days ago at No. 1 mine. The bumps there appear to be different from the one at Black Diamond or Carbonado. They have been attended by large outbursts of gas. At both Carbonado and Black Diamond the percentage of extraction is more than at Crow's Nest Pass; there, I have been told, it is only 8 or 10 per cent., still there have been bumps.

In the Cumberland mine there have been some bumps with a strong roof on a weaker floor. I remember one instance when the return airway was choked up overnight.

In each of the instances that I have noted there is the strong roof, comparatively strong floor and strong coal.

We have not been able to work out a solution for the bumps. At Black Diamond they became so severe under cover of about 2000 ft. that the men became frightened and the management thought it best to abandon the mine. A large amount of coal was left. I believe that if they had attempted longwall retreating, it would have solved part of the problem.

In Utah there was a bump condition in the Sunny Side mine, but it was between two faults with extra pressure, and as soon as the fault was crossed the bump ceased.

G. S. Rice.—If I recall correctly, in the Black Diamond mine there was involved a problem of natural folding. The mine was located in a pitching anticline.

G. W. Evans.—Yes, there was arching and faulting.

G. S. Rice.—In the case that Mr. Evans refers to at the Coal Creek mine, Crow's Nest Pass, B. C., my investigation of the question in 1916, as reported to the Minister

of Mines in British Columbia[4] indicated that the phenomena of "bumps" and "outbursts of gas" were quite distinct. They might occur coincidentally where bumps were the chief manifestations but their origin and effect is dissimilar. Bumps occur as already described in deep mining with strong roof and floor but where the coal is quite normal in its bedding. Bumps in coal mining are identical, in being caused by overlying ground movements, with "rock bursts" and "air blasts" that occur in deep mining of ore beds, or thick veins whether copper, iron or gold ore, but which usually occur only in pillar extraction. "Outbursts" as applied in coal mining mean instantaneous outbursts of gas which throw out coal with much dust violently from the face, as from a blast, in some cases a huge blast, throwing out thousands of tons of coal. Below a depth from the surface of about 1000 ft. their occurrence is independent of depth. They more usually occur, though not necessarily, in pitching or folded coal beds. My observation has been in studying the occurrences in various coal fields that they are due to geologic folding resulting in local thrusts after the coal has reached the stage of a true coal and in the subsequent folding, there has been a local lateral, perhaps somewhat twisting, movement of floor or roof crushing the coal. This has liberated occluded gases of the coal yet at the same time it appears to have produced a nearly impervious outer wall or shell of compacted coal perhaps sealed by the float dust carried by the gases. Crushing tests *in vacuo* have shown that some coals from mines subject to outbursts give off several volumes of gas per volume of coal. The Coal Creek (B. C.) coal gave three volumes. When the wall or "shell" or concentric "shells" are attacked by mining and sufficiently weakened, the highly compressed gas contained within, held under pressure in the previously crushed coal, bursts out violently. Fine coal dust always accompanies such outbursts; perhaps the gas may have been held under the pressure in almost liquefied state on the surface of the particle of the dust.

In mines subject to outbursts of gas, the coal ranges in rank from a high-volatile bituminous coal as at the Cassidy mine, Vancouver Island, B. C., to anthracite in South Wales. The kind of gas given off in the outbursts is not known definitely, because immediate sampling is not possible. If the volume of gas is great or continues to come from the outburst area for a considerable time there may be opportunity for sampling. Enough sampling has been done to make it known that in mines subject to outbursts, of Belgium, South Wales, England and British Columbia (Coal Creek and Cassidy mines), also in now abandoned mines near Dunmore, Alberta, the predominating outburst gas is methane; whereas in certain mines in southern France near Alais, and in upper Silesia, Germany, the gas is chiefly carbon dioxide.

As concerns the origin of the carbon dioxide, various theories have been put forward, such as reaction of acid waters on limestone and that the gases are from expired volcanism in the region, which have migrated to the coal. The writer has put forward[5] the theory, based on some unpublished preliminary laboratory tests of forcing a synthetic mixture of coal gases through highly compressed coal dust, that carbon dioxide was a residual gas after the methane, and other hydrocarbon gases of lighter density had slowly escaped through the ages from the nearly sealed area of crushed coal, leaving the heavier gases such as carbon dioxide behind. Although outbursts always occur in mines that are rated as "gassy," the mines are not always "very gassy." There are hundreds of very gassy mines throughout the world, some with enormous flows of gas, yet comparatively few mines, and none as yet developed in the United States, are subject to violent instantaneous outbursts of gas.

---

[4] Bumps and Outbursts of Gas in the Crow's Nest Pass Mines. Annual Report, British Columbia Dept. of Mines (1916).

[5] G S. Rice: Discussion. *Proc.* South Wales Inst. of Engrs. (1927) 32.

Returning to the question of bumps and outbursts at the Coal Creek mine, the system of mining to which Mr. Evans refers, by which only about 15 per cent. of the coal is taken out, was adopted by the company at the time of my investigations in 1916. Although some "bumping" continued until the overlying rocks had settled, the bumps became less and less severe, indicating that the method was successful. This does not refer to instantaneous gas outbursts, which are another and, in my opinion, a separate serious problem. Mr. Evans probably is not aware that there was no expectation of regarding the 15 per cent. extraction as final; the method is to block out the coal and eventually when certain assigned boundaries have been reached, to extract as much coal as possible by some form of retreating longwall.

The "outburst" problem, which had been so serious in certain mines in the Crow's Nest Pass field as to cause their abandonment, before the Coal Creek mines started, had not seriously troubled the No. 1 mine until about five or six years ago, although this mine had been extremely gassy. Then outbursts occurred in certain areas. I think it is to one of these that Mr. Evans referred. Through the courtesy of Mr. Canfield, general superintendent, I have kept in touch with the difficult conditions met with and until the recent outburst, by remarkably able handling, loss of life had been prevented.

H. G. Moulton.—Our Institute is greatly indebted to the officials of the British Empire Steel Corpn. for the time and effort which they have given to the preparation of papers and discussions for our meetings. Even during the period of depression which the steel and coal industries have been passing through, Mr. Herd and Mr. McCall have been willing to undertake the months of study required for the preparation of this paper. It is particularly pleasant to receive such whole-hearted cooperation from companies operating outside the United States, thus testifying to the international character of our organization.

Some comment has been made on the bumps that have occurred in the Cassidy mines on Vancouver Island, British Columbia. The difficulties encountered in the Cassidy seam appeared to be a combination of gas outbursts and bumps resulting from release of stress. The seam is folded and irregular in sections, and presents a typical case of accumulation of stress through folding and displacement.

I am in accord with Mr. Rice in feeling that bumps result from a combination of relations in strength between the coal on the one hand and the roofs and floors on the other; when coal seams with strong roofs and floors are mined at great depths, the pressures are not relieved regularly as the coal is taken out, but accumulate and are relieved over large areas. The pressures transmitted through a strong roof before its failure will result in stress in the solid coal, which in turn may cause bumps in the entries. In this event, it may be necessary to widen the entries so that the displaced coal will not fall upon the tracks, or it may even be necessary to drive the entries below the coal so as to keep them in stronger material.

The problem of bumps and the adjustment of mining methods to meet the resulting conditions has an economic aspect, particularly in the northwestern part of the United States and in British Columbia; coal mines have been operating with room-and-pillar methods for the purpose of obtaining coal at the lowest cost and under such circumstances commenced operations on a profitable basis. In many cases as the mines have become deeper the occurrence of bumps resulting from increased stress due to the depth or to geological conditions has indicated the necessity of changing to longwall methods of mining. Unfortunately this condition has been faced at a time when the competition of crude oil has caused a reduction in the price of coal and consequent decline in profits to a point where many properties which on technical grounds should now be changing their mining methods to meet the change in operating conditions, involving in many cases entire new layouts, are prevented by financial considerations from making the necessary changes.

The lesson from this is that if a mine which is pitching steeply and gives every indication of developing excessives pressures and consequent bumps as the operations are extended in depth does carry the promise of sufficient profit to permit a proper layout in the beginning with provision for longwall mining and with wide entries or entries underlying the coal, it had better be viewed as a non-commercial property and the coal left standing awaiting the possibilities of the future. It is quite probable that a good deal of the trouble from bumps in coal-mining operations has come from attempts to operate properties on too narrow a margin of profit to permit the proper layout for operation.

H. N. Eavenson, Pittsburgh, Pa.—There are a couple of mines in Harlan County, Kentucky, which are beginning to experience this same trouble, but conditions yet are not nearly so bad as in Mr. McCall's mine or as in the Western mines. In one mine the cover runs from 1500 to 2000 ft.; in the other, it will be about 2600 ft., but at present they have about 2000 ft. The roof is generally hard; usually sandstone or very strong sandy shale. Sometimes there is a heavy draw slate over the coal. The coal is hard and brittle, and the bottom, I think, is not quite as strong as the top. The trouble there manifests itself in large masses of coal, sometimes 4, 5 or 6 ft. deep, bursting out over the face of the room.

In both of those mines the seams are practically flat. The trouble occurs usually in advancing places in a concentrated system of working that has a large area open, and I think that relieves the pressure other than in the local conditions mentioned.

In the second mine, which is smaller, workings extend over a great deal more open territory than there is in the Nova Scotia mine, and there is a good deal of gas, which comes out from the coal and makes considerable trouble. I do not think that there has been any fatality yet, but eventually in that district some mines will be working in seams as much as 3000 ft. deep, and it may be possible that sometime in the future they will have more trouble of this sort unless a type of mining is laid out that will be a safeguard against it.

R. D. Hall.—To my mind these bumps arise from arch rather than from bending-moment stress. As the roof gives way the arch spans extends further and further. These extensions were not confined to the period of operation but were noticeable when the mine was idle prior to the replacement of room and pillar methods by retreating longwall.

It seems to me that with a sandstone roof there is almost rigid resistance up to a certain point; then the resistance is suddenly overborne, bringing to bear on the coal a load that is extremely severe. With slate or shale there will be greater deformation under stress. Consequently dynamic stresses will not be set up. I am not referring, of course, to such dynamic stresses, or shocks, as Mr. Rice describes. They probably exist at times in British Columbia mines.

Reference was made by Mr. McCall to a lowering of the bottom rock after its ascent and to the coal descending with it thus leaving the immense cavity illustrated in the article. I cannot conceive that the pavement of the seam would lower after it had once risen and would like to hear Mr. McCall's explanation as to the manner in which this becomes possible.

G. S. Rice.—The specific bump at Springhill that is described in detail is of unusual interest because not only were the immediate results of the bump and general conditions carefully recorded but subsequent excavation followed, and it is really a very unusual record. I do not remember seeing any similar case. To me it is an instance of a shock bump, which has been transmitted through the roof and coal to the floor,

causing that to buckle upward and thrust part of the pillar toward the semi-longwall face.

T. L. McCall.—Mr. Hall referred to arching action. The paper has taken up that point. Where the author speaks on the theory of bumps, he mentions arching action and takes it into account.

Regarding the bottom coming up and going down again, that is a wave motion. We are of the opinion that these bumps are caused by pressure and shocks setting up a wave motion in the roof, and in the pavement, but mostly in the pavement.

As an example of what the wave motion in the pavement can do, I can cite an instance when we were beginning the use of longwall. We had laid our conveyor pans up along the face. We had chocks built along behind the conveyor, parallel with the face—hardwood chocks, 4 ft. square—then a bump occurred. It picked up one of the packs and planted it on the top of the conveyor pans and left it there. The conveyor pan was about 12 in. off the floor level. There must have been a considerable wave motion to pick up the chock, move it 10 ft. and place it, without a stick disturbed, on top of the conveyor pan. That occurred before full development of the longwall system; later we found that bumps do not affect the longwall face badly.

We have had the tracks of roadways thrown uphill.

G. W. Evans.—That happened in Carbonado mine and also in Black Diamond.

T. L. McCall.—Was the coal displaced?

G. W. Evans.—The pillar on the lower side of the gangway was badly crushed. but not displaced, and the pillars above the gangways were crushed to a powder, In one instance the roof had come down for a distance of 800 feet.

T. L. McCall.—In connection with the bumps at Springhill, usually the roof is not disturbed. The reaction seems to come from the pavement. Occasionally the jar seems to loosen the roof and bring it down, but that is the exception rather than the rule. That is why we are able to widen our levels; we have been able to carry them 30 ft. wide with just a line of chocks for support.

R. D. Hall (written discussion).—Mr. Herd's admirable description of the bumps at Springhill give much food for thought regarding the action of the roof. It must be remembered that all roofs do not act alike, especially when the systems of mine extraction differ and are of varying completeness. In the main there are four kinds of roofs:

1. Roof where the solid part of the cover is so thin that it will fail from shear. When a mass of unconsolidated material, such as Pleistocene gravel or glacial till, lies on the thin solid part of the cover, the probability of failure of this kind is increased.

2. Roof that spans only narrow openings such as rooms or headings where the solid part of the cover is strong. Here the roof is not subject to so much bending moment that it will break at the surface over the pillar, no matter how shallow or how deep the coal seam may be.

The under layers of the roof may be separate from the upper layers or may become so by reason of differential stress, by expansion from heat or by the presence of material that is susceptible to expansion from moisture or oxidation. These under layers may break off and form the much discussed Gothic arch. This failure may possibly, and probably does, often reach the surface.

These preliminary acknowledgments are made necessary because if one does not recite faith in them, "orthodox" petrodynamists will believe they are being denied.

3. Roof that spans a large opening but that is relatively so thin that the tensile strength of the roof is inadequate to resist the tensile stresses. Accordingly the roof is torn over the pillar from the surface some short distance downward, and also for a

similar distance in the mid-span, but in this case from the bottom upward. Then the platelike roof[6] becomes so much weakened that its ability to support itself seems ended.

There seems nothing to prevent both fractures extending, the one over the pillar downward, that over the opening upward, but scarcely is the sound of the rending over than the roof quiets down. The thrust of the two opposing rectangular masses (for such a shape they seem rapidly assuming as they try to revolve on the edges of the pillars) prevents any further motion. Until the thrust causes the measures to shear horizontally the roof cannot fall. When it fails by thus shearing, it again becomes a beam, breaks rapidly again and once more chokes, this time with a lower point of thrust contact. At last, the roof fails enough to rest on the broken rock in the excavation and the subsidence is practically ended.

All this has been said in brief and irrelevant introduction for fear it will be said that the preliminaries have been forgotten and that the type of roof failure I have hitherto chosen to discuss I have now discarded for another entirely different. Nothing is discarded; in what follows there is merely a recognition of another of the many forms of roof failure, and indeed there are other forms than those described in this discussion. It is as if one would descant upon the Hebrew alphabet, only to be told that the Latin alphabet is universally used, and is the only means of expression, and that the Greek alphabet, which was the subject of previous dissertations, is now already recognized by the writer as nonexistent. Returning to the fourth roof type:

4. Extremely thick roof which spans a large opening such as is created by longwall or results from methodical pillar drawing of a panel or series of panels. Such roof is almost sure to fracture over the pillar if mining is continued, but in some instances such failure is delayed so long that it seems as if it would never occur. In fact, the failure may be delayed almost indefinitely if the cover is extremely thick, if the rocks have great compressive strength at right angles to the stress they have to sustain and if the strata do not slide readily on each other to permit failure.

It must be remembered that in an exteriorly loaded weightless beam the resistance to bending per unit of maximum tensile and compressive stress would increase as the square of its depth, but as the beam is not weightless, and as its weight is proportional to its depth, the tensile and compressive stresses in a beam loaded only with its own weight will decrease proportionately to the increase in its depth. Accordingly a thick beam is slow to break over the support as the stress is smaller than with a shallower beam.

When a beam sags, compressive stresses are set up in the upper part of the beam over the center of the opening, just as occurs in an arch. These stresses combine with beam loads, which become greater and greater as they recede from the vertical over the center of the opening and approach the verticals over the supports.

The line of forces thus established passes from the vertical over the opening along a curved line shaped like one-half of an arch down to the mine floor. It is commonly termed a "stress polygon," though it is, of course, not a true polygon but a continuous curve in the overburden. The structure which thus sets up a line of stress resembling an arch is by no means necessarily itself of arch shape. It might be better to break away from precedent and term the stress polygon a "stress curve." This curve, though it is shaped like an arch, may exist in a roof in which not a ton of rock has fallen. It is an invisible line like the axis of the earth. This must be clearly understood or the discussion of the stress curve may lead to erroneous conclusions.

As has been said, the failure of the roof over the pillar may be almost indefinitely delayed, and at Springhill it seems likely that as the depth rapidly increases with distance the day of such rupture may never arrive.

---

[6] The expression "platelike" is used to imply great horizontal extension in two directions and not to suggest thinness. The plate might be 100 to, say, 1000 ft. thick, or even more.

Fig. 13 shows the way in which the compressive stresses resulting from thrust, or indeed from bending, are compounded with the weight of the rock incrementally and how the increments of weight bend the line of stress, or stress curve, down toward the supports. With a large thrust the curve flattens, as can be seen from the figure in which the three curves are based on thrusts in the relation 1:2:4.

One is always in a quandary when searching for words to express entities that have never been described. One can invent a new word or strain the meaning of an old one. In the one case the word is difficult to remember and sounds pedantic and in the other case it is misleading. I shall risk censure and adopt the second course in this discussion. The underside, or innerside, of an arch is termed the "intrados," its upper or outerside is named the "extrados." Let us use these words quite erro-

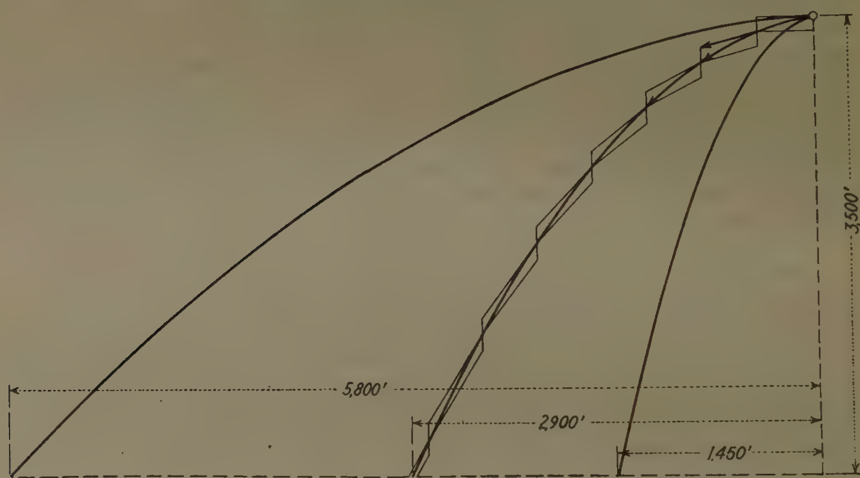


FIG. 13.—STRESS POLYGON, OR RATHER STRESS CURVE, WITH THRUSTS IN RATIO 1:2:4. STRESS IS REALLY TWO-DIMENSIONAL AND THE CURVE ONLY A TRACE.

neously in this discussion, regarding the intrados as the material of the roof which is below or within the stress curve and which has not fallen or sagged away from the main mass.

I know this is a confusing use of the word, for there is in most cases a real arch with a real inner or under surface which is truly entitled to be regarded as the intrados. The use suggested replaces a three-dimensional arch and its two-dimensional intrados by a two-dimensional stress curve and a three-dimensional intrados. Also I ask permission to use "extrados," for the rock mass above and beyond the stress curve, in an equally erroneous manner (see Fig. 14).

In a beam which has throughout equal elasticity the sum of the compressive forces may be regarded as concentrated at a point equal to one-sixth of the depth of the beam from the beam's upper surface. With any body of overburden, as the top rocks have much less strength than those further down, this point must be below the surface more than one-sixth of the depth of the overburden. Its exact location will vary with the nature of the surface rocks. Furthermore, equal elasticity throughout the mass cannot be safely predicated. The actual summation point for all the compressive stresses is hardly so important, however, as the point of maximum pressure and as the maximum stress that point sustains. Therefore it might be well to regard the line of forces or stress curve as the line of maximum stress, and also to designate the extrados as the masses above that line and the intrados as the adherent masses below it.

In a beam spanning an excavated area the lower half, if the beam is of equally elastic material throughout, is in tension. So also it is regarded as tending in that direction in an arch when the line of force is outside the middle third. Consequently the question whether the line of force or stress polygon should be considered as result-

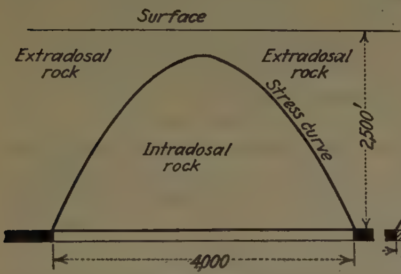


Fig. 14.—Stress curve over opening.

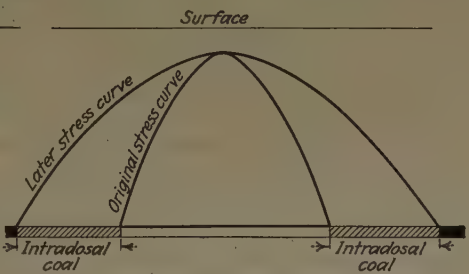


Fig. 15.—Stress curve after the face coal has become crushed.

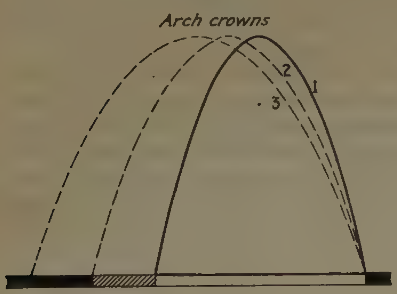


Fig. 16.—Shows how stress curve crowns move with excavation of coal or weakening of rib.

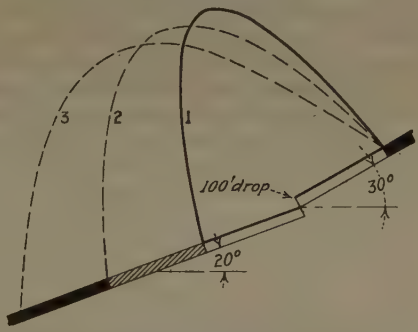


Fig. 17.—Same adapted to Springhill conditions.

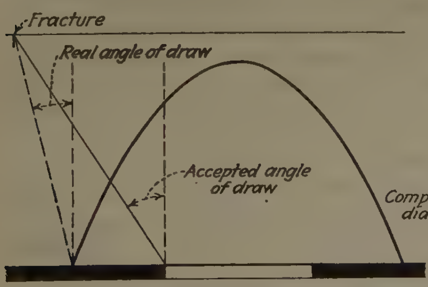


Fig. 18.—Shows the difference between the real and accepted "angle of draw."

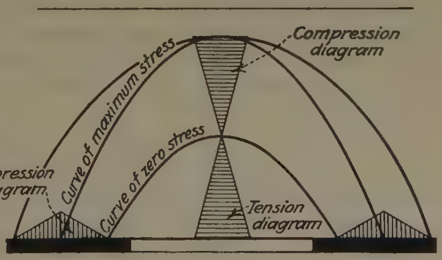


Fig. 19.—Suggests manner in which stress may be distributed in roof and on coal.

ing from the entire weight of the intradosal masses is an insistent one. Certainly it should not be so regarded in cases where the tensional stresses so destroy the strength of parts of the intradosal structure as to cause them to fall to the mine floor. But in some cases there will be only sagging. The mass thus separated will not necessarily fall.

The collapse of even these intradosal masses of mine roof is not by any means easy. The masses when ruptured will crowd each other, setting up thrusts and forming a new and separate stress curve, which must for its self-preservation necessarily be flatter than the superior stress curve, or curves. If it is not flatter, it cannot carry to the abutments and will fall. It attains that greater flatness either by reason of a greater thrust at the crown of the curve or by reason of lighter loading, the lightness, of course, being because much of the roof load is supported by one or more superior stress curves.

Whatever span needs to be covered by the stress curve will be covered by it or the roof will fail, but the span of the stress curve will never be one foot greater than necessary to reach adequate support, for it is only as the roof begins to fail that the compressive stress arises to support it. The stress is never a ton greater than will serve that need, so the thrust is always just that which will carry the stress curve over to adequate support. The roof will fail only when the rock or coal fails to sustain the needed stress and when it is no longer possible to shift the burden.

A true arch, one that has the shape of an arch and is loaded only by its own weight, has a stress curve that follows the line of a catenary. A stress curve in a mass of unbroken roof has the form of a parabola. The stress curve in a span that by "intradosal" caving has become arched has a form between a catenary and a parabola. It has a vertical loading that increases toward the haunches.

It is probable that the stress curve changes the location of its crown as the coal is mined away, or as the stress destroys the resistance of the pillar adjacent to the working face and so increases the area of non-resistance. With a level seam, and with mining in only one direction, if the destruction of pillar resistance is overlooked, the location of the crown should advance one-half as fast as the excavation. This movement of the crown also adds to the probability that fracture will be long delayed.

With a seam under 2500 ft. of solid rock, a span of a mile is by no means extraordinary. On the contrary it might be expected. Apparently at Springhill the excavated area and that area which has been weakened beyond resistance by bumps are together probably much more than 1 mile wide, though the upper "arch" limb must be much shorter than the lower, because the cover is much shallower.

It has been said already that where a beam is loaded with only its own weight the beam stresses in compression and tension will be inversely proportional to its depth. Thus with an overburden of 2500 ft. these stresses will be one twenty-fifth those where the overburden is only 100 ft. The denominator of this ratio will be greatly increased if the rotten surface rocks are deep. Thus if they are 20 ft. thick, the resistant overburden will be 2480 ft. in one case and 80 ft. in the other, and the stresses will be one thirty-first part in the former case of what they would be in the latter without regard to the weight of the rotten rocks, which would stress the thinner solid rocks 31 by 31 times, or 961 times as much as the thicker. With a stress curve taking the load, the tensile stresses would be still further reduced.

Deep mines, therefore, should have better roof than shallow ones. Springhill has such a roof. In deep roof, not only are the tensile stresses less, but as the cementing bodies are not leached out, the roof should be intrinsically stronger. Where the horizontal tensile stresses are less, the horizontal shears between layers of rock are less, and "drawroof" is less likely to form.

However, there is always the possibility of swelling ground and of rock naturally weak, and perhaps of cooling by ventilation or of heating when steam is introduced as for pumping, so one cannot always count on obtaining increased strength with depth. At some of the mines in the Glace Bay district the roof gets more troublesome as the coal becomes deeper. One can only explain that action by an assumption that the weakness is due either to expansible materials in the roof or natural incoherence, more likely the first.

Though what I expect to say later will show what terrific forces are set up by any mining operation that is conducted without caving, one cannot but be impressed from what Mr. Herd said regarding bumps, that the stress curve has only a limited area of incidence. When we talk of weight "riding the pillar" we mean not so much that the bending roof is crowding the coal pillar further and further, but that the point of stress-curve incidence, or rather line of incidence, is moving back over the coal. Consequently it would be just as well, as Mr. Herd rightly says ought to have been done at Springhill, to remove the coal from the area that is no longer competent to sustain the incidence of the stress curve and, therefore, no longer subject to that pressure.

Again, with apologies, I shall use the word "intrados" incorrectly and term this coal pillar, no longer subject to stress-curve pressure, the "intradosal pillar" (see Fig. 15). Sometimes in shallow measures an effort is made to restore the strength of the intradosal pillar, and this conceivably might be done to such a degree in shallow measures that the roof finding a resistance at that nearer point as great or greater than farther back might transfer the stress-curve to that point. In loose parlance, one might say that the roof is looking always for the nearest point of incidence competent to sustain it.

Perhaps one may be pardoned for suggesting that there might be a question whether the bump of Apr. 12, 1928, in Springhill mine really originated from the new retreating longwall, as the author seems to infer. Is it not conceivable that it was the result of pressure from the arch over the old extracted area? Might it not possibly have been the result of the stress curves of both systems intersecting? The pillars that fail at Springhill are perhaps at points where groined arches intersect.

That the cavity in the coal pillar and the extrusion in the roads run parallel to the line of the older extracted area and are at right angles to the advancing line of the retreating longwall seems to point to the predominant pressure as coming from the old waste rather than the new.

The faulting found in the mine might seem to be the result of the pressures at the foot of the stress curves extending over the new waste, but the faulting was doubtless earlier than the opening of the mine and was probably even prehistoric. It must have been found in part when driving the levels or Mr. Herd would certainly have referred to it as a possible evidence of recent pressure.

However, Mr. Herd's argument for the wave action radiating from the retreating longwall is entitled to great weight. Mr. Herd is disposed to believe that the locus of the maximum pressure was what is now at the end of the cavity nearest the coal face of the longwall retreat. In the language I have employed, it would be the footing of the stress curve. If Mr. Herd's assumption is correct the action might be thus stated: When the rock broke under the stress, the coal was forced out, and when the incidence of the stress curve shifted to a point farther back to find coal that would oppose the stress, the stress was relieved and the floor fell back, causing a cavity to form.

Though at Springhill there has been no surface rupture over the pillar, that is, no "draw," it might be well, nevertheless, to say that where the fall of the roof is delayed so that the stress curves withdraw from the face, the angle of draw—if draw should ever occur—should not in level beds be measured from the vertical over the coal face backward to the line of rupture, but if possible from the vertical over the footing of the stress curve to that same surface fracture. This will explain why the draw is sometimes so remote from the vertical over the true face.

Leaving coal intradosally may support intradosal roof masses and reduce the vertical components in the stress curve and thus enable the width of the span of that curve to be increased without collapse, so that the weight will move back. Perhaps this is why the leaving of pillars in imperfect coal recovery causes the weight to "ride

the pillars." In saying this I am thinking of other mines than Springhill, where the pillars have been ridden apparently in part because of the loss of pillar coal in room and pillar workings.

An instance of 1121 ft. of draw is derived from Manchester, England, where the coal was 11 ft. thick and the depth of seam 1725 ft., making the apparent "angle of draw" 33° 1′. Here the ratio of subsidence between 1903 and 1906 was only 27.8 per cent. The actual subsidence was 3.06 ft. At this mine, judging from the subsidence ratio, either the pack walling was unusually complete, close and competent to support the roof, or pillars had been left in. The subsidence was even less between 1900 and 1903, for the record shows that it was only 0.10 per cent. Evidently there was delayed caving, on account of the great depth of the coal, or there would have been more subsidence between 1900 and 1903, but there must have been intradosal support or the final subsidence after the draw fracture occurred might have been expected to exceed 27.8 per cent. Thus, two reasons—failure of intradosal pillars to sustain the arch and intradosal support of internal rock masses, the latter thus relieving the arch of some weight—caused the pressure of the stress curve probably to extend well back of the face before the roof collapsed, and the draw, therefore, should be measured from the vertical over the edge of the truly *resistant* coal back to the line of surface rupture. This instance is taken from a review of the minutes of evidence of the Royal Commission on Mining Subsidence.[7]

In the above I have calculated the "angle of draw" but must point out that it may be only a mathematical abstraction. The proof that there really is a fracture line lying at any such angle has never been satisfactorily presented. There is a fracture above and one near the pillar below, but that they join is yet to be proved It is unfortunate that the expression "line of draw" has been coined. It gives the honorable status of a proven fact to what is only an extremely questionable conjecture.

Before attempting any calculations of the stresses in the roof and pillars arising from large extracted areas such as those at Springhill, it would be well to consider in general the strength of rock and coal under compression. All tests of the strength of these materials are, and of necessity must be, fallacious, because both must be much stronger in large masses than in small specimens, for in the former instance lateral extrusion or sliding on oblique fracture planes is prevented.

The Scranton Engineers' Club committee found from its experiments at Lehigh University, made I believe in 1900, that the strength of anthracite varied roughly as the square root of the ratio, which the width of the square block being tested bore to the height. If a specimen had a 2 by 2-in. cross-section and the height was 1 in., its ratio would be 2, and if it had a height of 1 in. and an 18 by 18-in. cross-section, the ratio would be 18. In that case the ratio between the ratios would be 9.1 and the relative strengths would be 3:1. If the coal in the specimen 2 by 2 by 1 in. had a strength of 10,525 lb. per. sq. in., such as was exhibited by coal of the Clark seam at Hazleton No. 5, the coal in a specimen 18 by 18 by 1 in. would have a strength of 31, 575 lb. The experiments were all on pieces running from 2 by 2 by 1 to 2 by 2 by 4 in.; so one cannot say just how far the analogy could be pressed. Certainly not to the limit, for then a pillar infinitely large could be exposed to infinite pressure without crushing. But even if it did crush, it still would have strength, for fine sand after it has been consolidated by pressure makes a sustaining pillar.

In passing it may be said that the weakest of all the anthracite tested in blocks measuring 2 by 2 by 1 in., of which the 1 in. was the depth, failed only under a pressure of 3025 lb. per sq. in. That anthracite came from the Little Lykens Valley seam at Lykens.

---

[7] S. W. Price: Mining Subsidence,. A Review of the Minutes of Evidence of the Royal Commission on Mining Subsidence. *Iron & Coal Tr. Rev.* (1928) **117**, 872.

The coal at Springhill is hard and perhaps it might be reasonable to put its strength in pillars 100 ft. wide and even longer at 23,000 lb. per sq. in. I am well aware that in assuming such a figure one exposes oneself to much criticism. The answer can only be that, regardless of all the results of the testing laboratories, the strength exhibited by certain strong mine pillars, and the concentration of the stresses over a small area of incidence, makes this assumption necessary, and in fact the laboratories themselves have propounded a theory that if it may be carried to its limits would make such a strength low. It must be remembered that the coal at Springhill is 10 ft. thick and the pillars are 100 ft. wide, so the ratio is 10:1, which compares with the ratio of a specimen 2 by 2 by 1 in. as 5:1. The square roots will compare almost as 2.236:1. With coal as strong as Hazleton No. 5 this would give a strength of 23,534 lb. per sq. in. Widening the pillars should measurably increase this pillar strength.

W. J. Macquorn Rankine, quoting an address by Mr. Fairbairn before the Manchester Philosophical Society, puts the strength of strong Yorkshire sandstone (mean of 9 experiments) at 9824 lb. per sq. in. He does not say what were the dimensions of the specimens tested. It might be safe to assume that sandstone has a strength of, at least, 60,000 lb. per sq. in. in unbroken bed, though it must be remembered that the maximum stress is near the surface of the ground and the pressure is not on the bed but along the planes of bedding where exfoliation is easy, the more so as the pressure is not uniform but increases toward the outside layer. However, Mr. Rankine quotes Mr. Fairbairn as saying that his experiments showed that the resistance of strong sandstone to crushing in a direction parallel to the layers is "only" six-sevenths of the resistance to crushing in a direction perpendicular to the layers.

In order not to introduce complications in the calculations of the stresses of the stress curve, let the coal be assumed to be level and let a section of the arch 1 ft. wide be taken. Let P = weight of the overburden over the entire span. Let h be the height of the overburden and s the span of the curve described by the center of inertia of the compressive stress, which span must be wider than the opening in the coal by half the width of the area over which the stress falls and may be more if some of the coal has already been crushed and its resistance destroyed. Let w be the weight of the overburden per cubic foot. Then $P = hsw$.

Let T be the total thrust of the stress curve at the median line. Let h' = the maximum height of the stress curve, namely the height at this same median line. The curve described is constructed to represent the entire thrust as located at its center of inertia.

Then by the rule of the parabola

$$\frac{\frac{1}{2}P}{T} = \frac{h'}{\frac{1}{4}s} = \frac{4h'}{s}$$

$$T = \frac{hs^2w}{8h'}$$

h' is always less than h, for the center of inertia of the stress curve is below the upper resisting layer perhaps one-sixth of the depth of overburden plus the depth to this resisting layer, which possibly is 25 ft. or so. Overlooking the latter, $h' = 0.833\,h$ provided the modulus of elasticity for rock is equal both for extension and compression.

Then

$$T = \frac{hs^2w}{6.67h} = \frac{s^2w}{6.67}$$

If the specific weight of the overburden is 160 lb. per sq. ft.

$$T = \frac{s^2 \times 160}{6.67} = 24s^2$$

Suppose s = 5280 ft., then

$$T = 669{,}081{,}600 \text{ lb.}$$

The pressure is zero at the neutral axis and two-thirds of the maximum at the center of inertia. Thrust is exhibited from the neutral axis to the outer resistant layer, which for the nonce has been assumed as the surface, though it is probably at least 25 ft. lower. Let the distance over which thrust is exerted be a.

The average stress per sq. ft. is

$$\frac{T}{a} = \frac{T}{\frac{1}{2}h}$$

If the height is 2500 ft., the average stress is 535,265 lb. per sq. ft. and the maximum stress must be 1,070,530 lb. per sq. ft. or 7434 pounds per square inches.

Evidently the maximum resistance of the rock is nowhere near reached. The roof evidently can extend over a much larger excavated area and can spread itself over a much more extensive bumped area if the haunches will stand the strain. Note also how the depth of the overburden h is eliminated in the calculations of thrust, for the thrust would be inversely proportional to the height if it were not for the fact that the load is proportional to that same factor. Height would actually *reduce* the compressive strain if the structure were weightless and externally loaded. Thus because the overburden is thick is no reason for assuming that the total thrust is severe. It is rather a reason for expecting that the thrust per square inch will be low for any given span.

Still taking the thickness of the overburden as 2500 ft. and the span as 5280 ft. the half weight of the overburden will be 2500 by 160 by 2640 or 1,056,000,000 lb. Combining the weight of the overburden with the thrust; *viz.*, 669, 081, 600 lb. geometrically, the stress will be 1,249,122,000 lb., and the angle of the stress to the vertical will be 32° 21.5′. The maximum stress will be put at 23,000 lb. per sq. in. The average stress will be half that, or 11,500 lb. per sq. in. or 1,656,000 lb. per sq. ft. Thus the stress will dispose itself over a distance of 755 feet.

If this is true, the bump is due to a local giving way at the line of maximum stress. The coal intradosal to this maximum stress is thus squeezed intradosally and the coal will be extruded into the heading intradosal to the maximum stress. The pressure continuing down into the floor will cause the floor to heave in any near-by extradosal heading; this would be explained by the difference in action in the two headings. The inclination of the stress would make such extradosal effects more likely and more extended.

Misfires in Anthracite Coal Mines

By T. D. Thomas,* Lansford, Pa.

(New York Meeting, February, 1929)

In this paper, major attention is given to misfires in mines where electric multiple shot-firing is the system used.

Misfires are sometimes caused by one action or condition and at other times by a combination of two or more. They may be divided into two classes: (1) those caused by workers in the mines, (2) mechanical or manufacturer's causes. The second cause will be discussed first.

Misfires Caused by Faulty Detonators

The important part of an electric blasting cap, to the user where multiple shot-firing is done, as in the steep-pitch, thick-vein, anthracite mines, is the bridge wire. In single shot-firing this factor is not apparent, but in mines where multiple firing is done any variation in the resistance of the bridge wire, where full or even rated battery capacity is required, will most likely cause a misfire. The reason for this is obvious. If the resistance in the bridge wires is not the same, some of the detonators will fire because of their greater comparative resistance. Variance in resistance of bridge wires is caused by a slight difference in the diameter of the bridge wire, also by a difference in the composition of the bridge wires. Again, where the bridge wires have become rusted, there will be a variance in resistance.

About the year 1922, one of the large coal companies was greatly alarmed by the number of serious and fatal accidents caused by misfires. The general manager issued an order that these accidents must be greatly reduced, and if possible, entirely prevented.

The officials at the mines had to determine the various causes of misfires and have them corrected. The rock contractors who use large quantities of 60 to 80 per cent. explosives and do all firing electrically were thoroughly instructed as to the proper methods of doing their work, and closer supervision was given to their work. The manufacturer's representatives stressed the importance of the use of galvanometers; accordingly these were purchased and used. Larger firing lines were installed under the supervision of a demonstrator of the company manufacturing the caps and explosives. However, misfires again occurred and it was evident that the source of the trouble was not wholly

* Mine Inspector, Lehigh Coal & Navigation Co.

caused by the men at the mines. The miners' representatives were requested to run tests of their electric shot-firing equipment. This included tests on firing batteries and instantaneous and delay detonators.

The tests were made under the following conditions: Length of firing line from firing battery to leg wires, 105 ft. of No. 18 gage wire; all resistance readings of circuit made on Wheatstone bridge; all shots to be fired by one person. In this test two different types of firing batteries and also two different electric caps were used. The caps are called *X* and *Y* and the firing batteries, *M* and *N*.

TABLE 1.—*Tests of Electric Shot-firing Equipment*

Test No.	Cap No.	Make of Cap	Firing Battery	Resistance, Ohms	Shot No. (*F* = Fired; *M* = Missed) 1 2 3 4 5 6 7 8 9 10 11 12 13 14 15 16
1	6	*Y*	*M*	11	*F F F F F F*
2	6	*X*	*M*	11	*F M F M F F*
					Cap 2 in test 2 placed in hole 1
3	6	*X*	*M*	11	*F F M M F F* Cap 4 in test 2 placed in hole 3
4	10	*Y*	*M*	16.8	*F F F F F F F F F F*
5	15	*Y*	*M*	23.5	*F F F F F F F F F F F F M F F*
6	10	*X*	*M*	16.5	*M M M M M M M M F F*
7	8	*X*	*N*	13.8	*M M M M M M M M* 8 caps which missed in test 6 used in Test 7
8	8	*X*	*M*	13.8	*F F F F F F F F* 8 caps which missed in Test 7
9	1	*Y*	*M*		1 (Fired through 12 ohms)—Fired
10	1	*Y*	*M*		1 (Fired through 70 ohms)—Fired
11	1	*Y*	*M*		1 (Fired through 60 ohms)—Fired
12	1	*Y*	*M*		1 (Fired through 55 ohms)—Fired
13	1	*X*	*M*		1 (Fired through 55 ohms)—Missed
14	1	*X*	*M*		1 (Fired through 50 ohms)
15	1	*X*	*M*		1 (Fired through 45 ohms)
16	1	*X*	*M* (new)		1 (Fired through 40 ohms)—Fired
17	1	*Y*	*M*		1 (Fired through 75 ohms)
18	1	*Y*	*M*		1 (Fired through 70 ohms)
19		*X*	*M*		1 (Fired through 70 ohms)—Missed
			M		1 (Fired through 65 ohms)
			M		1 (Fired through 55 ohms)—Fired
20	1	*X*			Which failed once; shows 1.2 ohms; connections O. K.
21	1	*X*			*M* which failed twice; shows 1.14 ohms; connections O. K.
22	1	*Y*			*M* which failed in test 15; 13 holes; shows 1.08 ohms; O. K.
23	1	*Y*			*N* Fired at 15 ohms.
24	1	*Y*			*N* Fired at 15 ohms.
25	1	*Y*			*N* Fired at 16 ohms.
26	16	*Y*	*M*[a]	24.5	*F F F F F F F F F F F M F F F F*
27	16	*Y*	*M*	24.5	*F F F F F F F F F F F F F F F F*
28	15	*X*	*M*	24.5	*F F F F F F M F M F M F F F F F*
29	3	*X*	*M*	13.8	*M M M* (combination of *X* and *Y*
	5	*Y*			*F F F F F* caps.)
30	3	*X*	*M*		*F F F* (these failed in test 29).

[a] Firing battery from mine in use 18 months.

The deductions from the foregoing tests with electrical multiple shot-firing are that the *M* firing battery was more reliable than the *N* type. The tests also indicated that the *Y* detonators were more positive than the *X* type, which showed variance in resistance.

The manufacturers selling *X* detonators were at this time selling another electric blasting cap which did not have the same resistance in the bridge wire as its standard *X* detonator. When both types of detonators were used in one series of shots, misfires occurred. The manufacturer disposing of these detonators certainly should have investigated the bridge wires rather than have the various mines get into trouble through misfires.

TABLE 2.—*Tests with Detonators of Two Makes Together*

Test No.	Cap No.	Make of Cap	Firing Battery	Resistance, Ohms	Shot No. (*F* = Fired; *M* = Missed) 1	2	3	4	5	6	7	8	9	10
31	3	*X* (8 ft.)	*M*		*F*	*M*		*M*						
	3	*X*[1] (4 ft.)	*M*				*M*		*M*	*M*				
32	2	*X* (8 ft.)				*F*		*M*	(This test of 5 holes missed in test 3.)					
	3	*X*[1] (4 ft.)	*M*				*F*		*F*	*M*				
33	1	*X* (8 ft.)						*M*						
	1	*X*[1] (4 ft.)	*M*							*F*				
34	4	*X* (8 ft.)			*F*	*F*								
		X[1] (4 ft.)	*N*				*F*	*M*						

Tests 31 to 34, given in Table 2, were made to show that in multiple firing detonators made by different firms should not be used in the same series, as misfires will occur through variations in resistance. Immediately after these tests were made, the maker of *X* and *X′* detonators removed all *X′* caps. The best method in this respect is to have only one manufacturer's exploders at an operation.

After the introduction of delays in multiple shot-firing, until recently, the time fuses were cut and the exploder inserted on the fuse by the miners; caps were furnished in containers containing 25 caps. Complaint was received by many companies from coal consumers about caps being found in their coal bins. To prevent this, all delays were made at the factories of *X* and *Y* firms. Many more misfires occurred and further investigations followed.

There were few complaints on the *Y* new type, but on the *X* new type complaints were made and sustained. It was found that the trouble was caused by the waterproofing compound seeping through about the cap and solidifying above the detonating compound. The electrical component and the fuse attached functioned, but the spark from the fuse did not get to the compound in the detonator. This is another example of misfire due to manufacture.

MISFIRES FROM INSENSITIVE EXPLOSIVES

Occasionally, misfires are caused through insensitiveness in permissible explosives. These explosives if kept in storage at the mines too long absorb moisture and become insensitive; and some shipments

received from the factories a short time before our tests also showed insensitiveness. The standard method of determining the sensitiveness of permissibles was used in making these tests. Some permissibles would not explode at 4 in. separation. (Separation tests were made of the explosives that would not fire at 4 in.). The difficulties experienced by the makers may not be fully realized by the writer; however, we as users of permissibles feel that insensitive explosive should not be received, thus exposing our miners to a hazard that is not the fault of the company purchasing or the miners using the explosives.

A detonator was shown recently of which the chief merit was said to be that fewer misfires will occur with it than with other detonators, as this detonator has power enough to detonate insensitive powders. This makes it rather evident that the manufacturer recognizes the fact of insensitive explosives.

Misfires from Man Causes at the Mines

Our investigations regarding the human element in misfires showed that the lack of knowledge on the part of users related to:

1. Method of making proper connections;
2. Method of supporting leading and leg wires;
3. Use of wire of too small a gage;
4. Improper turning of firing battery;
5. Improper care of firing battery;
6. Not cutting fuse as shown and instructed;
7. Improper placement of fuse in detonators;
8. Not reporting defective firing batteries;
9. Shooting from the trolley wire.

1. *Making Connections.*—Many of the miners in making connections on the leg and leading wires did not take sufficient time to make a proper connection; in fact, many connections are merely hooked, therefore the current generated by the firing battery is not conducted to the detonator and misfires result. The assistant foremen and firebosses while inspecting the working places can observe such defects and have them remedied.

2. *Supporting Wires.*—Until recently, it was almost impossible to force the miners or rock contractors to support the wires properly, the usual custom being to drive nails in the legs or collars and wrap the leading wires about the nails, thereby breaking the insulation on the wires and causing a ground. There might be 30 such supports on one firing wire. A method of avoiding this practice is to make supports in the mine shop from waste material. The miners spike them to the legs or collars, thus keeping the wires free from contacts. In rock work the contractor drills a 2-in. hole 4 in. deep at intervals of 20 to 25 ft. A wooden plug is driven into the hole and a support is attached to the plug.

Misfires have been greatly reduced by this method and the miners soon realized its advantages in regard to their own safety.

3. *Wire Gage.*—With the introduction of electrical shot-firing in the mines, No. 18 gage wire was generally used, regardless of distance or the number of holes to be fired. It was difficult to induce the miners to purchase other wire because it increased costs, which they claimed was a violation of the agreement. At one mine a rock contractor was having many missed holes, which retarded driving of the chute. He was also losing considerable money on missed holes. The officials at the mines finally convinced him that a larger gage and rubber-covered wire would reduce the trouble. After installation of the larger wire, using the same *Y* detonator and *M* battery, much better results were obtained. The success in this instance soon spread around the mines. The miners sent a committee to the superintendent with a request that he order the same type and size of firing wires for them, saying that they would gladly pay the additional cost.

4. *Turning Battery*—For many years the miners had an idea that it was necessary to warm up the firing battery before giving it the final twist. Unfortunately, this procedure was frequently gone through, after all connections had been made. Sometimes, in a series of seven or eight shots, three holes were exploded and five holes would be reported as misfires. It required much argument to convince the miners that the proper method of twisting the battery is to get the generator up to its highest speed.

5. *Care of Battery.*—Improper care of firing batteries is the cause of many misfires. Frequently they are taken into the mines and left where they can absorb moisture and fine particles of dust, and their use is continued until misfires become too general. The miner always blames the manufacturer for misfires, but in such instances the detonators are always tested, and nine times out of ten the misfires are found to be caused by the miner. At many operations the miners are required to bring their firing batteries to the assistant foreman's offices at the end of every shift, where they are kept in a dry, warm room overnight, when all the moisture absorbed by the battery during the day has been taken out. At these offices small testing devices have been installed. The test block consists of twelve 2.6-volt incandescent bulbs. If a miner complains that the battery is not firing up to capacity, the assistant foreman makes the test in his presence. If it is defective, a good battery is furnished the miner and the other is sent to the electrical repair shop.

6. *Cutting Fuse.*—This cause of misfires has been eliminated by having the manufacturers make all delays complete at their factories.

7. *Placing Fuse.*—This cause also has been eliminated in the same way.

8. *Reporting Defective Batteries.*—Miners fail, sometimes, to report defective firing batteries. To avoid defective installation some com-

panies require that firing batteries be sent to the electrical repair shop every six months to be thoroughly tested and have necessary repairs made.

9. *Shooting from Trolley Wire.*—The foregoing causes led to the practice of shooting from the trolley wire, which caused a number of fatal and serious accidents. This method of firing is greatly condemned by the U. S. Bureau of Mines.

Regulations Regarding Misfires

Misfires are reported by the miner in whose working place the misfire occurred to the assistant foreman, who records the misfire in a book kept for that purpose. A danger board is placed at the entrance of the chamber and no person is allowed to enter until the following day. The assistant foreman notifies the fireboss as to the locale of misfires, and the latter inspects them while making the morning inspection. No misfires may be drilled out. The rules specify this: "No shot that has missed fire shall be drilled out. Drill a new hole in such a manner that it will not come in contact with the explosives in the original hole. Second hole must be charged lightly."

Serious and fatal accidents have been caused by violating this rule. To avoid violating the rule the assistant foreman will deputize a reliable miner, if in doubt as to the reliability of the miners in whose chamber the misfire occurred, to see that the drilling of the new hole is done in compliance with the rules and regulations. The misfire when discharged is reported to the assistant foreman, who again records in the book the method of firing the misfire. The rule in reference to returning to a misfire is not the same as that adopted by the majority of mines using electric shot-firing devices, in which a reasonable amount of time is to elapse before returning to a misfire. The extension of the time to the following day was not so much because of fear of hang fires, but with the fixed purpose in mind of having the miners use more care in preparing their shots to avoid misfires.

Fatalities Resulting from Misfires

Following are some details of four serious accidents in anthracite mines resulting from misfires, arranged chronologically.

Drilling Out a Shot

Feb. 13, 1925.—A contract miner, working in a breast, was fatally injured by an explosion of a permissible dynamite which he was drilling out of a missed hole. The hole was one of four which he and his buddy had charged on Feb. 10. Only one had exploded, leaving three missed holes in the face of the breast. An investigation as to the cause of the accident

was made by the State Mine Inspector, district superintendent, colliery superintendent, foreman and company mine inspector. It was noted that four holes had been drilled, one being 2 ft. from the west pillar, and one 4 ft. from the west pillar; the other two were drilled relatively in the same points from the east pillar. The only hole that detonated was that 4 ft. from the west pillar, which was not a successful shot, being more like a blow-out shot.

The question naturally arose: Why three missed holes out of four at one face? In order to answer this question several determinations were made, as follows:

1. Was a firing battery of sufficient capacity furnished?

Answer: The battery used by the miner was a 50-hole type owned by a rock contractor and was in good condition and capable of detonating 4 holes.

2. Was all material used in blasting suitable for the work?

Answer: At this place were eight time fuses, which were all, by electrical test, found to be satisfactory as each was fired with a small firing battery. A box containing 50 time-exploders was obtained from the supply and tested as follows:

Test No.

1. 1 time fuse, end *not cut,* fired, did not flash at end.
2. 1 time fuse, end *cut,* fired, did flash at end.
3. 1 time fuse, end *not cut,* fired, small flash at end.
4. Battery through rheostat capable of firing 10 holes.
5. 1 time fuse, end *not cut,* fired, did not flash.
6. 5 time fuse, ends *not cut,* all fired, but only one flashed.
7. 8 time fuse, ends *not cut,* all fired, but only one flashed.

The significance of this list shows that while the material furnished is good it will not perform the work unless the ends on the delay fuses are cut squarely, not slanted, so that the powder flash will be in close contact with the fulminate in the cap.

3. How then can missed holes be caused when materials are good?

Answer: Careless handling, which may be divided into five causes:

(*a*) *Injuring Leg Wires.*—On certain instantaneous detonators there are two wires known as legs, soldered to a small platinum wire, set within the fulminate, above which is placed a composition to hold the wires in place. The same construction is followed in delays, except that fulminate is eliminated and regular fuse is used, the powder in the fuse being ignited by the heat from the bridge wire; in both, however, care must be used in handling and especially while tamping, otherwise one or both wires may be broken or pulled out. In tamping, provided that the correct type of tamper is not used, the insulation on the wires may be torn and positive and negative wire come in contact, causing a short circuit.

To eliminate the damage to insulation on leg wires, how should the tamper be made? The Anthracite Code states (see General Rules, No. 30): This will allow wooden tampers to be used. If wood, copper or other soft material is used, the diameter at the face of the tamper should be the same diameter as the cartridge, and a groove made from the face backward to the handle affording sufficient clearance for both leg wires, and rounded to prevent the tearing of the insulation. Wires or legs should be held in position in the groove while tamping.

(*b*) *Improper Connections.*—Connections are frequently loosely made and current is not conducted.

(*c*) *Wrapping Wires around Nails.*—The wire is wrapped tightly about a nail; the insulation broken and a short circuit is produced.

(*d*) *Insufficient Battery Capacity.*—Miners generally leave a firing battery for long periods in moist places. This deteriorates considerably the capacity of the battery so that it will not produce sufficient current to heat the bridge wire and thus causes a missed hole.

(*e*) *Deterioration of Explosives.*—Keeping explosives too long in damp places causes deterioration and will cause missed holes. (See General Rules, No. 26, sufficient for one day's work.)

It has been shown by the seven tests that the material and electric detonators were good; test No. 4 showed firing battery to be right; the fact that the battery used by the miner is operated in shooting on rock gangway work by the rock contractor eliminates the question. The foreman and assistant foreman then put up new firing wires and attached them to the shots; they tried to fire with a 20-hole battery and 50-hole battery but failed each time.

These tests then will lead to but one solution, which is careless handling, producing either a torn leg wire from the bridge or torn insulation from wire causing a short circuit, or ends not properly cut in fuses.

As the holes missed fire, what should have been the procedure?

Answer: The miners should report all missed holes to the level foreman, who will record the same in the report book and see that the place is fenced off. It evidently will be a policy of safety and produce an economy to have either the level foreman, assistant foreman, or some reliable miner enter the place with the miners to see that the shot is not withdrawn or the hole reopened. (See General Rules, No. 31.)

The accident was avoidable had rule 31 been obeyed.

Reopening Missed Holes

A contract miner working in a chute was seriously injured and sent to the hospital, Aug. 17, 1926. He had a missed hole and instead of doing as the Mining Law (Article XII, Rule 31) directs, he endeavored to reopen the hole, with the result that the charge exploded and injured him about the face and hands. It was only because the coal is soft

(gob) that the man was not killed. This was the second accident of this nature within three weeks. It is evidently necessary to use some method to stop this practice of reopening missed holes.

A contract miner working in the second crosscut was injured about the face and hands by a shot which he detonated while drilling out a missed hole on March 9, 1928. He had drilled and charged two holes in the crosscut, using one instantaneous and one delay detonator, there being a short period between the firing of the shots. He knew that both holes did not fire. As he knew he had a missed hole he should have placed a danger board at the manway of the breast, then notified the level foreman, who would record the missed hole in the book kept for such purpose, and keep men from entering the place for 12 hr. The miner did not do so, but went up the breast shortly after the hole missed and commenced to reopen the hole. (see Article XI, rule 31.) Had the law in reference to missed holes been obeyed the accident could have been avoided.

Drilling into a Hole

A contract miner was seriously injured on April 6, 1927. From the evidence it appears that he drilled into a missed hole, not accidentally but deliberately, and in so doing ignited the explosive; the blasted coal hit him about the body. He was taken to the hospital where his case was pronounced serious, but not likely to prove fatal. The method used by the miner was to start a hole about 18 in. from the missed hole, directed to intersect the missed hole about 24 in. from the mouth. This was in violation of all instructions and good judgment. It evidently seems necessary that all laws pertaining to missed holes should be complied with and, in addition, that no missed hole be touched except under the direction of an assistant foreman or a competent deputy foreman.

(For discussion of this paper, see page 237.)

Misfires in Bituminous Coal Mines

By W. H. Forbes,* Grant Town, W. Va.

(New York Meeting, February, 1929)

As permissible explosives and electric detonators are now generally recognized as the only safe means of blasting in coal mines, this paper is limited to their use.

Causes of Misfires

There are many different causes of misfires, but the principal ones are as follows:

1. Explosives which have deteriorated through improper storage either at the storage magazine, the distributing magazine, or at the working face. This is not only true of the explosive itself, but also of its detonating agent.
2. Improper priming and charging at the working face. This commonly occurs when the explosives are handled by inexperienced persons.
3. Improper methods of tamping; the leg wires of the detonator may be abraded or broken by the tamping bar or stick, thereby causing a short or broken circuit.
4. Faulty leading wire, or electrical apparatus of insufficient strength to explode the detonator.

Prevention of Misfires

Storage is an important factor, especially with permissibles of a nitrate or ammonia base. Both explosives and detonators should be stored in substantially built, fire-resisting magazines and should be situated so that the minimum amount of moisture and heat is present. They should be so constructed that a constant supply of fresh air enters; the floor of the magazine should be raised to permit the circulation of air under the stored explosives. Boxes should be stored top side up and in such a manner that the old explosives are used before those of a later delivery. This eliminates chance of deterioration from lengthy storage. Detonators should be similarly handled and stored.

The distributing magazine stock should be regulated so that not more than one day's supply is on hand.

---

* Safety Engineer, New England Fuel & Transportation Co.

No shot-firer or miner should be permitted to take inside more than is necessary for one day's work. While in the mine it should be stored in dry containers, placed in special cubby holes or boxes. If for any reason the supply taken into the mine has not been used, it should be returned to the magazine at the end of the shift. This also applies to detonators.

Proper Methods of Priming

Although there has been some controversy regarding the correct method of placing the detonator, all leading manufacturers of explosives agree that the best results are obtained when the detonator is placed to point toward the bulk of the charge. This can be accomplished by placing the detonator in the first cartridge entering the hole, with the detonator pointing to the mouth of the hole, or by placing the detonator in the last cartridge entering the hole, with the detonator pointing toward the back of the hole.

Several methods are used in priming cartridges, the most common being that of placing the detonator in the end of the cartridge and making its position secure by looping the leg wires of the detonators around the cartridge or threading them through the cartridge.

Charging

The following practical rules should be observed in charging shot-holes:

1. None but competent and experienced shot-firers should be permitted to carry and handle detonators, and charge or fire shots.
2. Before any explosives are charged into a bore-hole, all dust or drillings must be removed from the hole.
3. When placing a charge of explosives in a hole care should be taken not to abrade or kink the leg wires of the detonator.
4. An explosive charge, regardless of the number of cartridges used, should be pushed gently to the back of the hole, all at one time. Avoid force. Press firmly with a wooden tamping bar and tamp solidly to the collar of the hole.
5. Part of a cartridge should be placed on the opposite end of the charge from the primer with the closed end of this part cartridge next to the full cartridge; in other words, the open end of the part cartridge which may have absorbed moisture should either be placed against the coal at the back of the hole or against the stemming, to avoid any possibility of the part stick interfering with complete detonation if it has become insensitive through absorption of moisture.
6. Damp clay or earth, moist sand or rock dust should be used for stemming, never coal dust or rock chips. Care should again be taken not to kink the leg wires.

Firing

1. Be sure that all wire ends that are to be connected are scraped bright and clean.

2. Leading cable of sufficient size and in good condition should be used. Care should be taken of the connection of the leg wires with the leading wires. A battery or electrical device of sufficient strength (permissible type) should be used. This is important, because many misfires are the result of batteries of insufficient strength.

Experience at Federal Mines

At Federal mines 1 and 3 of the New England Fuel & Transportation Co. at Grant Town and Everettville, W. Va., approximately 1400 shots are fired daily; but few misfires have occurred, and these were traceable to the causes mentioned. When tetryl No. 6 electric blasting caps were used not a single complaint was registered, but when fulminate caps were used numerous complaints of misfires immediately came to our attention.

Although it is true that a few of the misfires were not traceable to defects in the fulminate type of electric blasting cap, our own experience leads the writer to believe that the tetryl type is superior and more reliable than the fulminate type. This statement may make it appear that the writer is partial to the tetryl type of electric blasting cap, and while this may be true, this conclusion was arrived at not only from our experience in connection with the use of both types over a considerable period, but also from comparative tests which the writer had conducted on the surface.

It has been the writer's experience that little or no consideration is given to the purchase of explosive supplies by many coal-mining officials. Price rather than quality is the major consideration—in other words, dollars and cents rather than execution and safety. If more officials would interest themselves enough to have comparative tests conducted, it is possible that more attention would be given to quality and safety than to price.

The tests which the writer had conducted by the various makers of electric blasting caps were on relative strength and water-resisting qualities. There is the well-known lead-plate test, in which the strength of one is compared to the other by the shape and size of the perforation; in addition to this we carried on an insensitive powder test.

In all tests the tetryl cap proved its superiority over the fulminate type. In these tests, 20 caps of each type, taken from the regular stock, were used.

In the water-resisting tests, the caps were subjected to immersion in water 10 min. at 50 lb. pressure. All tetryl caps fired, but all fulminate caps failed. An additional test of tetryl caps immersed in water 30 min.

under the same conditions all fired, but the fulminate type all failed again after having been immersed in water for a 10-min. period only.

Although this may at first appear to have no bearing on misfires, what the writer is attempting to bring out is that there is less danger of misfires when the tetryl type is employed, especially when explosives may be slightly deteriorated by storage either at the magazine or inside the mine.

Handling Misfires

Some State mining laws specify how misfires are to be handled. The writer recommends these methods:

1. Allow a period of at least 10 min. to elapse from the last attempt to fire the shot before approaching the misfired hole. After disconnecting the leading wires from the battery, short-circuit the ends by twisting, then wind or reel the leading wire. On reaching the face, immediately disconnect the leading wires from the leg wires of the detonator.

2. By short-circuiting the ends of the leading cable and reeling it, possibility of a premature explosion from stray currents is eliminated.

3. Do not permit any miner to drill out a misfired hole. To meet with the requirements of the State mining law and the company rules, we require the miner to drill another hole not less than 12 in. from the one that has misfired. This hole must be charged lightly and after firing, and when the coal is being loaded out, a careful watch is kept for the unexploded detonator and explosive.

4. Do not permit a shot-firer or miner to use any detonator or explosive that has failed to explode in any other hole. The shot-firers should be required to keep a careful check of all misfires and report them to the officials in charge.

Company Rules

The New England Fuel & Transportation Co. has laid down the following rules for the handling of explosives:

1. Explosives must be carried into the mine in proper containers (electrically nonconducting) and stored in cubby hole in the rib at least 100 ft. from the face.

2. Detonators must be carried into the mine in leather containers. These are locked before they are taken out of the supply magazine and must remain locked until the shot-firer reaches his section.

3. All unused explosives and detonators must be returned to the magazine at the close of each shift, where the detonators are checked against the number of cap checks turned in by each shot-firer.

4. None but experienced and competent shot-firers are allowed to handle detonators or charge and fire shots.

5. Shot-firers must use at least 100 ft. of leading cable and must be out of direct line of shot.

6. Not more than one shot may be fired at a time.

7. Shot-firers must see to it that holes are not bored on the solid before charging them.

8. Clay, earth, or rock dust must be used for stemming, to extend from the explosives to the collar of the hole.

9. Primers must be made by placing the detonator in the center of the cartridge parallel to its length, and must be placed in the bore-hole so that the detonator is pointing toward the bulk of the explosive charge.

10. When part of a cartridge is used in connection with one or more cartridges the open end must be placed against the coal at the back of the hole, or if the primer has been placed at the back of the hole the open end of the part cartridge must be placed against the stemming.

11. Wooden tamping bars only must be used to tamp bore-holes.

12. Permissible approved shot-firing devices only may be used to fire shots.

13. Shot-firers must examine places for gas before and after each shot is fired.

14. Shot-firers must not fire a shot in any place until the roof, ribs, and floor have been thoroughly washed down between the face and the rock-dusted area, or as far back as the hose will reach.

15. Shot-firers must see to it that all places are properly posted, including a "safety post," before shots are fired.

16. Care should be taken to prevent the leading cable from coming in contact with rails, pipe lines, or mine cars when shots are being fired.

17. Shot-firers must see to it that all persons are at a safe distance, and shall give proper warning twice—"Fire in the hole"—before each shot is fired.

18. All bug dust must be removed from undercut before shots are fired.

19. Foremen and shot-firers must see to it that all new miners are provided with a copy of company shooting standards before they are permitted to start work.

Conclusions

Briefly, the writer would say that misfires would be few and far between if the following practices are carried out in connection with the handling of explosives and electric blasting caps.

1. Have comparative test conducted by various explosive manufacturers. Purchase and use only explosives and detonators best adapted for the particular blasting.

2. Provide proper storage magazines and see to it that the old stock is moved out and used before that of later deliveries. (Permissible

explosives and electric blasting caps are perishable goods and should be treated as such.)

3. Have detonators handled by experienced and competent shot-firers only. Similarily, all shots should be tamped and fired by shot-firers. (Blasting should be considered highly skilled labor, and should be done by skilled workmen.)

4. Provide suitable material for stemming at convenient places, and eliminate the possibility that miners may use fine coal or slate chips, etc.

5. Provide suitable leading cables and reliable shot-firing devices.

6. Mine officials should be thoroughly acquainted with all phases of storing and handling explosives at their various operations.

7. Maintain the same strict supervision in connection with the charging and firing of shots that is usually given to other phases of mining.

8. Prohibit the promiscuous storage of explosives and detonators underground by employees.

9. As miners are employed, provide each of them with a copy of company shooting standards. (Diagram the place where holes should be drilled, and insist upon compliance with standards.)

While in all probability our shooting standards and rules pertaining to the care and handling of explosives are not fully complied with, nevertheless most satisfactory progress has been made since their adoption and not a single accident has occurred.

(For discussion of this paper, see page 237.)

Misfires in Non-metallic Mining (Limestone)

By A. W. Worthington,* Pittsburgh, Pa.

(New York Meeting, February, 1929)

It would be futile in this short discussion to attempt to cover the subject of misfires with the thoroughness which it deserves. No effort is made to set forth a list of the many causes of misfires, nor to analyze the various preventives and methods of treatment which are advocated by those who are most interested in promoting good practice. These details, as well as the fundamentals, have been rather fully covered in many published articles based on years of experience in the manufacture and use of explosives and on research covering specific problems. We shall assume the knowledge of and familiarity with those many facts which have been proved beyond reasonable argument, and will discuss the problem of misfires as it has actually been experienced in the operation of the limestone quarries and underground mines of the Pittsburgh Limestone Co. and its associated companies, which produce fluxing stone for the furnaces of the United States Steel Corpn. in the Pittsburgh district.

These opinions are based on data compiled from written records at each of our operations, and they are concurred in by our assistant general manager in charge of operations, our safety director, and the superintendents at each of our plants, to whom we are indebted for assistance in gathering many of the data. Each of these men is not only interested in the subject but is fully capable of analyzing results obtained under his supervision.

The entire subject of the proper storage and use of explosives, of which the problem of misfires is only a part, has been given a great deal of study. In this we have been aided by having the advantage of the publications of the U. S. Bureau of Mines. A number of the states have also, more recently, given to quarrying and general mining methods the attention which they deserve. Their findings should prove of value to the operators. We feel, further, that operators particularly owe gratitude to the various manufacturers of explosives for their thorough and intelligent study of the many problems relating not only to the manufacture but to the use of explosives, and for their willingness and efforts to disseminate the results of their investigations.

---

* Assistant to General Manager, Pittsburgh Limestone Co.

Importance of Explicit Rules

We believe it to be most important that the details of each type of operation be carefully considered to the end that the issuance of rules unnecessarily dogmatic in form or substance may be avoided. Such rules may have the advantage of being concise and may cover admirably certain conditions known intimately to their authors but taken literally they may in fact defeat their purpose by actually creating hazards in a type of operation differing, perhaps only slightly, from the one originally intended to be covered. The operators and their employees should neither be advised in written technical articles nor required by rule or regulation to comply with instructions which do not clearly define the field intended to be covered, and which have not been given the thought and care necessary to insure their being reasonable and applicable to that entire field. We realize the difficulty of avoiding inconsistencies, but we believe that their elimination is essential in order that experienced men who are actually engaged in the use of explosives shall not be confronted with the necessity of willfully breaking a rule which they know, if complied with, would add to the hazards of their work. Such a procedure can only lead to loss of confidence by these men in the rules and in the agency that created them—whether that agency be their employer, the manufacturer of explosives, or a state or federal bureau.

The writer has in mind many such rules, some of them having grown out of coal-mining practices; others appear to have been compiled after only casual thought, but with the sincere intent to prevent some particular bad practice. In the solving of the problem of misfires as well as of other dangers incident to the use of dynamite, the writer believes that this subject is so important that a few examples will be mentioned. An amusing one, but one having nothing to do with misfires, was related by another quarry operator. He claimed that the regulations of the state in which his plant was located provided that as a warning before each blast, "A steam whistle shall be blown," and that an inspector, quoting this wording, refused to allow the use of an electric siren. This is an extreme case and indicates, perhaps, poor judgment on the part of the inspector, but it shows just as clearly that little thought was given to the original phraseology.

A rule that is often printed and quoted and sometimes promulgated as a regulation is the one requiring, without alternatives, that in the event of a misfire, a second hole shall be drilled parallel to and not nearer than 2 ft. from the first. Under certain conditions, this may offer the best solution; for instance, where the stone surrounding the misfire is so shattered by near-by shots that the misfire is suspected but not definitely known, or where the shattered rock blocks access to the hole in which the misfire occurred. It is our opinion that this method of treating misfires is, in

any of our several types of limestone operations, one of the most dangerous that we could pursue. In open quarry work the danger in drilling the second hole near a churn-drill hole 50 to 100 ft. deep, or near an air-drill hole 15 to 50 ft. deep, becomes evident, particularly if the stone is seamy or likely to contain caves, or is a measure having its various ledges lying in a marked and perhaps varying dip. This danger becomes further increased if such holes have been sprung. In snake holes the method is practically impossible. It is true that in quoting this rule its advocates occasionally add the warning, "After shooting the second hole, careful search should be made for dynamite and primers from the first hole, which may not have been detonated by the second blast." We contend, even though the second hole could be drilled in comparative safety, that rarely would the explosive in the first hole be detonated by the second one and anyone who has attempted to hunt for a live primer and a large quantity of explosive in a pile of broken stone, containing perhaps thousands of tons, would elect to choose almost any other method of remedying the misfire. To go to the other extreme, no one, we believe, would suggest drilling a second hole 2 ft. from a misfired block hole which had been drilled in the center of a 2 or 3-ft. block. In underground limestone mining, with six or eight converging face holes, some horizontal and some inclined, it again becomes dangerous to attempt to drill a hole near enough to one which has misfired to offer any chance of detonating the first by firing the second charge. If it is not detonated, it will either be thrown out into the room in the pile of broken stone, or will remain in the face in a mass of shattered rock, leaving conditions worse than before.

Another rule often cited is the one specifying, "No hole shall be charged with more than one kind of explosive." This we believe to be needlessly restrictive. It certainly can not be objectionable, and it is often advisable to use a high-strength dynamite as a primer to shoot either black powder or dynamites of lower strength or of less sensitivity. If the rule was intended to prohibit the use of a squib and black powder in firing a dynamite charge, it should not have been difficult to make the meaning clear.

One further example of rules which we believe to have no place in the proper operating of limestone mines or quarries is that one providing, "All holes shall be tamped to the collar of the hole." Compliance is obviously unnecessary in deep primary holes in quarry operations. In the shallower holes in both quarries and mines, the use of more than an effective amount of stemming increases the liability of injury to or disturbance of the caps, wires, or fuse during the longer tamping operation. If a misfire occurs, the existence of the excess stemming creates a condition less favorable for remedial measures.

In the effort to eliminate all of the ills incident to the use of explosives, the book which 30 years ago was to be found in every home under the

title of "The Family Doctor" should be remembered. It, too, prescribed definite remedies for every known ill, gave little consideration to changing conditions and attempted in a few words to replace the necessity of study and training in medicine and surgery. The avoidance of this unsound method of procedure is essential, as it undermines the fundamentals in the education of the men who are actually using the explosives and, after all, in this education lies the only hope of securing materially better practice.

Description of Operations

In order that the findings from our experience may be of any value, it will be necessary to outline as briefly as possible the scope and type of our operations. They have been conducted over a period of more than 30 years in 21 different limestone and dolomite properties in the western and central parts of Pennsylvania, in Maryland, and in the eastern portion of West Virginia. Five years ago we were producing stone at 10 quarries and 5 mines. As the result of development of the capacity of the most favorable properties, the company has been able gradually to reduce the number of plants, so that now only seven—four quarries and three mines—are being operated, without sacrifice of capacity.

Of the quarries, one in western Pennsylvania, with a capacity of about 4000 gross tons in 8 hr., produces limestone from the Vanport vein, which is a practically horizontal measure about 20 ft. thick. Two faces are operated, due to variation in analysis, the top one being about 15 ft. and the bottom one 5 ft. high. Some years ago, at a time when the entire face was shot in one blast, churn-drills were used in part but at present we are using air drills in each of the ledges. In primary shooting, 1½ by 8-in. gelatin dynamite and electric blasting caps are fired by 220-volt, 60-cycle alternating current. From 50 to 200 holes are connected in parallel series, using No. 10 or 12 lead wires. Blasting caps and fuse are used in block-hole shooting at the power shovels. Clay is used as stemming in all holes.

The other three open quarries, each having a capacity of about 1200 tons in 8 hr., are in central Pennsylvania and the eastern part of West Virginia. In these the face varies from 50 to 100 ft. in height. The greater part of the primary drilling is done with churn drills, although tripod air drills are used to some extent. These three quarries at present are not equipped with crushing plants so that there is, consequently, a large amount of block-hole shooting required, the drilling being done with jackhammers and the holes shot by cap and fuse. In the churn-drill holes, gelatin dynamite is fired by either cordeau or electric blasting caps. Primary holes drilled with tripod air drills in one quarry are sprung and loaded with black powder; in the other two quarries, with quarry gelatin without springing. At one quarry a blasting machine

is utilized; at the other two, 220-volt, 60-cycle alternating current is used. Clay, loam and sand are used as stemming.

In the three underground limestone mines in western Pennsylvania, with a daily capacity of 2000 to 4000 tons each, the operating methods are so similar that it will be necessary to give only one outline. The limestone, practically horizontal, is approximately 20 ft. thick. Room and pillar mining is pursued with rooms about 40 ft. wide and pillars 20 ft. thick. There is left in place about 3 ft. of limestone as a roof and 1 ft. as floor, removing the 15-ft. face by angle shooting. In the vertical face, three or four 1½-in. holes are drilled horizontally, or nearly so, in a vertical row along each side of the angular block to be removed. These holes converge to a vertical line at a depth of about 12 ft. During the 5-year period covered by the data given later, we have used both gelatin and ammonia dynamites. In all of this primary shooting, electric blasting caps are fired either by blasting machine or by 220-volt, 60-cycle alternating current. The size of lead wires varies from No. 10 to No. 18 annunciator wire. For stemming, dummy cartridges are prepared of ordinary wrapping paper and either fine limestone dust, obtained at the crusher, or clay. The stone is loaded, for the most part, by air shovels or electric shovels. On account of the necessity of continuing some hand loading and avoiding delays to the shovels, there is a considerable amount of block-hole shooting required. For this duty, electric blasting caps are used whenever, owing to dampness or other adverse conditions, the probability of a misfire would make the use of fuse inadvisable.

Number of Misfires

Bulletin 288, the latest publication of the U. S. Bureau of Mines covering quarry accidents in the United States (those for 1926) shows that there were, in 1700 quarries of all types during that year, a total of 162 lost-time accidents per thousand 300-day workers and in limiting the types of these operations to limestone quarries, the rate is increased to 200 per 1000. During that same year our companies were operating four limestone quarries and five limestone mines in which 1718 (300-day) workers were employed. Our lost-time accident rate was approximately 40 per 1000, or about one-fifth of the rate for all limestone quarries in the United States. During that year we had but one lost-time accident caused by explosives; this gave a rate of 0.58 per 1000 workers, which is about one-thirteenth of the corresponding average rate for all limestone quarries in the United States. During the year 1928, with 1446 (300-day) workers, our lost-time accident rate was only 10 per 1000 workers. There were no accidents from the use of explosives.

In the study of misfires it early becomes evident that there are few if any published data to indicate the number of misfires which occur in

SAFETY THE FIRST CONSIDERATION

PITTSBURGH LIMESTONE COMPANY
SHOOTERS DAILY REPORT

TO ______________________ SUPERINTENDENT ______________________ QUARRY ______________ 192....

Location in Quarry Section	Primary Holes			Secondary Shots Number	Sprung Holes Number	Snake Holes Number	Seam Shots Number	Bench Holes Number	Misfires		Feet of Fuse	Ft. of Cordeau		Pounds of Black Powder	Pounds of Dynamite					Electric Blasting Caps Designate Tetryl or Fulminate by T or F				Number of Blasting Caps	Estimated Gross Tons of Stone	Was Quarry Face Inspected After Shooting
	Number and Depth	Month Drilled	Time Shot						Primary Shots	Secondary Shots		Plain	Countered		5x16 50%	1½x8 40%	1x8 35%	5x16 30%	Special	No. 6 Quantity	No. 6 Length	No. 8 Quantity	No. 8 Length			
TOTAL																										Total Tonnage

EXPLOSIVES ON HAND AT DISTRIBUTING MAGAZINE AT END OF DAY
Location
Location
Location
Location
Location

TIME OF SHOOTING—SECONDARY SHOTS			
First Round	M	M	M
Second Round	M	M	M
Third Round	M	M	M
Extra Round	M	M	M

CHECK NUMBER AND NAME		
First Shooter	Check Number	Name
Head Inspector	Check Number	Name
Well Driller	Check Number	Name
Tripod Driller	Check Number	Name

NOTE:—

For Quantity of Explosives on Hand At Main Magazine Refer to Storekeepers Report.
In case of Misfire, the Following Data Should be Given on Reverse Side:—Power Circuit, Battery Wire Connections, Data Relative to Burden, Depth of Holes and Sketch.
Estimators Data of Primary Shots to be Given on the Back of this Report.

BLASTING MUST BE DONE IN COMPLIANCE WITH THE LAW AND COMPANY RULES

FIG. 1.

SAFETY THE FIRST CONSIDERATION

PITTSBURGH LIMESTONE COMPANY
SHOOTERS. DAILY REPORT

TO________________SUPERINTENDENT ________________MINE ....192....

LOCATION		Number of Primary Holes	Primary Drillers Check Number	Number of Angles			Block Holes		Number of Holes		Electric Blasting Caps Designate Tetryl or Fulminate By T or F				Number of Blasting Caps	Feet of Fuse	Pounds of Dynamite			Estimate Gross Tons Stone	MISFIRES		Number of Stuck Holes	Was Mine Face And Roof Inspected After Shooting
											No. 6		No. 8											
Entry	Room			Left	Center	Right	Total Number	Shot Firers Check No.	Roof	Bench	Quantity	Length	Quantity	Length			30%	40%	Special		Primary	Secondary		
TOTAL																								
EXPLOSIVES ON HAND AT DISTRIBUTING MAGAZINE AT END OF DAY																								
Location																								
Location																								
Location																								
Location																								
Location																								

CHECK NUMBER AND NAME.

First Shooter

Inspector

TIME

Entered Mine

First Shot

Last Shot

Left Mine

NOTE:—

For Quantity of Explosives on Hand At Main Magazine Refer to Storekeepers Report.
In Case of Misfire, the Following Data Should be Given on Reverse Side:—Power Circuit, Battery Wire Connections, Data Relative to Burden, Depth of Holes and Sketch.
Estimators Data of Primary Shots to be Given on the Back of this Report.

BLASTING MUST BE DONE IN COMPLIANCE WITH THE LAW AND COMPANY RULES

FIG. 2.

various types of operations as compared to the number of holes fired. About five years ago we installed at each of our plants a system requiring daily reports from shooters, by which accurate record has been kept showing the position of all primary shots, total number of all holes, caps, feet of fuse and the pounds and strength of the explosives used. (These are reproduced as Figs. 1 and 2.) These report blanks provide space also for a description of all misfires and of the facts disclosed by the investigation to determine their causes.

From these reports, in an attempt to obtain fundamental data for this paper, a record has been compiled of all misfires during the operation of the four quarries and three mines which are active at the present time.

TABLE 1.—*Frequency of Misfires*

Quantities	5 Years—(1924–28 Incl.)						1 Year—1928						Per Cent.
	Four Quarries (Total)		Three Mines (Total)		Grand Total		Four Quarries (Total)		Three Mines (Total)		Grand Total		
Holes fired:													
Cordeau		1,112		0		1,112		525		0		525	
Electric blasting caps		286,734		1,159,545		1,446,279		54,245		326,122		380,367	
Cap and fuse		2,084,973		370,988		2,455,961		533,085		112,811		645,896	
Total		2,372,819		1,530,533		3,903,352		587,855		438,933		1,026,788	
Explosives used, lb.		4,056,187		8,979,178		13,035,365		699,710		1,743,450		2,443,160	
Misfires (all types): Ratio = Total holes fired / Total misfires	Misfires[a]	Holes per Misfire	Misfires[a]	Holes per Misfire	Misfires[a]	Holes per Misfire	Misfires[a]	Holes per Misfire	Misfires[a]	Holes per Misfire	Misfires[a]	Holes per Misfire	
Holes fired per misfire:													
Cordeau	1	1,112	0	X	1	1,112	0		0	X	0		
Electric blasting caps	16	17,920	367	3,160	383	3,780	1	54,245	93	3,510	94	4,050	9
Cap and fuse	177	11,780	232	1,600	409	6,000	20	26,650	22	5,130	42	15,380	212
Average	194	12,230	599	2,560	793	4,920	21	27,990	115	3,820	136	7,550	72
Ratio = Total lb. of explosives used / Total misfires		Lb. per Misfire		Lb. per Misfire		Lb. per Misfire		Lb. per Misfire		Lb. per Misfire		Lb. per Misfire	
	194	20,900	599	15,000	793	16,400	21	33,300	115	15,200	136	18,000	12
Average dynamite in each cap and fuse shot, lb.		0.29		0.48		0.32		0.27		0.52		0.30	

Last column indicates the better performance during 1928 by giving the percentage of increase in 1928 grand totals over the grand totals of the preceding 4-year period, 1924–27.

[a] Misfires—each attempt to fire one or more holes, resulting in failure to detonate all of the explosive in all of the holes is considered to be one misfire. With electric blasting caps or cordeau, each misfire may include one or more holes. With cap and fuse, each misfired hole is one misfire.

TABLE 2.—*Number of Misfires and Their Causes*

Causes	5 Years (1924–28 Incl.)			1 year.—1928			
	Four Quarries	Three Mines	Total	Four Quarries	Three Mines	Total	Expectancy
Cordeau:							
Damaged cordeau (old stock)......	1	0	1	0	0	0	0
Total holes fired...............	1,112	0	1,112	525	0	525	525
Electric blasting caps:							
Defective wiring in circuit (various faults)........................	11	113	124	1	26	27	36
Defective electric blasting caps (manufacture)................	1	114	115	0	35	35	31
Defective blasting machine (maintenance)........................	0	26	26	0	1	1	10
Insensitive dynamite or weak caps..	0	19	19	0	15	15	2
Wet conditions at hole (dynamite or caps)........................	0	14	14	0	5	5	4
Faulty operation of blasting machine........................	0	8	8	0	1	1	3
Damaged leg wire (tamping)......	0	3	3	0	0	0	1
Detached portion of explosive remaining undetonated, in cave formation after firing churn-drill hole........................	1	0	1	0	0	0	0
Unknown (probably defective wiring or caps)..................	3	70	73	0	10	10	24
Total number of misfires (electric blasting caps).............	16	367	383	1	93	94	111
Total holes fired with electric blasting caps................	286,734	1,159,545	1,446,279	54,245	326,122	380,367	380,367
Cap and fuse:							
Damp fuse (absorbed dampness too readily after loading)...........	0	152	152	0	3	3	65
Damp fuse (absorbed dampness too readily before loading)..........	1	17	18	0	8	8	4
Wet fuse (sudden rains and melting snow)........................	125	0	125	15	0	15	39
Fuse cut by flying spawl from nearby shot........................	18	27	45	1	5	6	16
Fuse kinked—powder train broken	15	0	15	0	0	0	5
Insensitive dynamite or weak caps (manufacture)................	2	9	11	1	3	4	3
Faulty crimping.................	9	0	9	0	0	0	3
Defective blasting caps...........	0	8	8	0	0	0	3
Failure to light fuse..............	0	4	4	0	0	0	1
Cap too near bottom of hole.......	3	0	3	3	0	3	0
Drop of oil in powder train of fuse during manufacture...........	2	0	2	0	0	0	1
Powder train of fuse broken (manufacturing).....................	1	0	1	0	0	0	0
Cap pulled from charge by a too large tamping stick...........	1	0	1	0	0	0	0
Unknown.......................	0	15	15	0	3	3	5
Total misfires—cap and fuse.....	177	232	409	20	22	42	145
Total holes fired with cap and fuse	2,084,973	370,988	2,455,961	533,085	112,811	645,896	645,896

Last column shows the expectancy of total number of misfires from various causes, which would have occurred in 1928, had the rate $\frac{\text{(Holes fired)}}{\text{(Misfires)}}$ remained the same as was experienced during the preceding 4-year period (1924–1927, incl.)

These are all 5-year records, with the exception of those for one quarry and one mine, which are for only 2 years and 2½ years respectively. At these seven plants during this 5-year period, there were fired approximately 4,000,000 holes and 13,000,000 lb. of explosives. Table 1 is an analysis of these quantities to determine the frequency of misfires. Another study is made of the causes of all of the misfires occurring during this period and the grouping has been made in Table 2 to show the number of misfires from each cause.

Frequency of Misfires

It will be noted in Table 1 that there is a separation made between the totals of the four quarries and of the three mines, showing the number of holes fired in the quarries as 2,372,819 and in the three mines as 1,530,533, or a grand total of 3,903,352 holes of all types. Further subdivisions, according to methods of firing, show the following grand totals for number of holes fired: cordeau, 1112; electric blasting caps, 1,446,279; cap and fuse, 2,455,961. All of the holes fired with cordeau were, of course, in the quarries. Of the holes fired with electric blasting caps, 80 per cent. were in the mines and of those fired by cap and fuse, 85 per cent. were in the quarries. Of the 13,035,365 lb. of explosives, 69 per cent. was used in the mines.

During this 5-year period at seven plants, there occurred one misfire with cordeau, 383 with electric blasting caps, and 409 with cap and fuse, for a total of 793 misfires. From a comparison with the total number of holes fired by each method, we find that the frequency, as measured by the number of holes fired per misfire, is: cordeau, 1112; electric blasting caps, 3780; cap and fuse, 6000; weighted average, 4920 holes fired per misfire. Another comparison shows that one misfire occurred for each 16,400 lb. of explosives used. In the last four columns of Table 1, similar data are given showing the performance at these same plants during the year 1928, in which there were 1,026,788 holes fired and 2,443,160 lb. of explosives used. A comparison of the records of 1928 with the average of the preceding four years shows a considerably better performance during 1928. For cordeau, there were no misfires in 525 holes; electric blasting caps, 4050 holes per misfire, or a gain of 9 per cent.; cap and fuse, 15,380 holes per misfire, or a gain of 212 per cent.; weighted average, 7550 holes per misfire, or a gain of 72 per cent. For each misfire experienced 18,000 lb. of explosives was used, or a gain of 12 per cent. over the preceding 4-year period.

In the preparation of these data, a misfire was considered as being each attempt to fire one or more holes resulting in the failure to detonate all of the explosive in all of the holes; with cordeau or electric blasting caps each misfire, therefore, may include one or more holes. With electric blasting caps, it was usual, fortunately, that two or more adjoining

holes failed in each misfire. In most shots a safer condition, and one more easily recognized and remedied, results when a series of such holes, rather than only one hole, misfires. In shooting with cap and fuse, each misfired hole is considered as one misfire.

Because we have been able to find no similar data in the United States, it is difficult to be certain that our frequency ratios are good ones. For the purposes of our own company, we can only compare the performances of the different plants. Inquiry was made of a number of other engineers and operators who, without any definite data to guide them, variously estimated the probable number of misfires in the fields familiar to them as 0.5 to 1.5 per cent. The lower percentage would represent, of course, one misfire for each 200 holes fired. This, when compared with our performance of 1928 of one misfire for each 7550 holes, would seem to indicate that the estimates of these other engineers were unduly pessimistic. In *The Colliery Guardian* for Aug. 3, 1928, we find that similar computations have been made for the coal mines of England, Wales and Scotland during the year 1927. They show that in about 35,000,000 shots with electric blasting caps, there were nearly 19,000 misfires, for a ratio of 1880 holes per misfire, and that in 13,000,000 shots with cap and fuse there were about 7000 misfires, for a ratio of 1841 holes per misfire. Table 1, for our limestone operations during the year 1928, shows that the number of holes per misfire for electric blasting caps was more than twice and for cap and fuse shots more than eight times the number obtained in Great Britain. It is realized that the conditions may be quite different and that we are not sufficiently familiar at this time with the British method of making these calculations to draw a true comparison. We may, however, have offered a yardstick of some worth for future studies of the subject.

Causes of Misfires

In this analysis (Table 2) are listed the various causes of all misfires experienced at four quarries and three mines during the 5-year period 1924–28, inclusive. These causes and the number of misfires due to them have been further subdivided to show those which occurred in blasts fired by cordeau, electric blasting caps, and cap and fuse. The total number of holes of each type is also shown. A similar grouping is made for the misfires occurring during the year 1928. In column 8, there is shown our natural expectancy of the number of misfires due to each of the various causes had they continued at the same rate during 1928 as was experienced during the preceding 4 years.

It will be seen that with cordeau, the one misfire in 1112 holes during the 5-year period was due to the use of old stock. During February, after the preparation of Table 2, we experienced at one of the quarries another misfire in which one hole failed in a blast of 13 holes. Cordeau

was used in the holes and trunk line. The top portion of the missed hole was destroyed in the blast, so that it has not been possible to determine whether the failure was due to defective cordeau or to a poor connection.

With electric caps, it will be noted that nine causes of misfires are listed. In the 5-year period, about 32 per cent. were due to various faults in the wiring of the circuit; 19 per cent. are grouped as unknown—the evidence, in most instances, having been destroyed by the firing of adjacent holes. It is probable that the majority of these were due to defective wiring or caps. The misfires ascribed to defective electric blasting caps or to insensitive dynamite or weak caps constitute, during the 5-year period 35 per cent., and during 1928, 53 per cent. of the total number of misfires, the remainder being due either to our own faulty workmanship or to causes unavoidable within a reasonable degree of precaution. Perhaps we should have assumed an even greater percentage of misfires to our workmanship, as in some of those attributed to defective caps it is possible that other causes were the true ones. For those shown as being insensitive dynamite or weak caps, we only know that the cap fired but did not detonate the explosive. The cap manufacturer might rightly claim that the misfire was due to insensitive dynamite and, in turn, be met with the possibility of having furnished an occasional cap which was not of full strength. In the grouping of misfires due to each of these causes, our superintendents could only use their best judgment after considering all the attendant circumstances.

In blasting with cap and fuse, 13 causes of misfires are shown. In the 5-year period, 72 per cent., and in 1928, 62 per cent. of all misfires in this method of shooting were attributed to either damp or wet fuse, and about 11 per cent. to the fuse being cut by flying spalls from a neighboring shot. Only about 5 per cent. is attributed to defective explosives. We have made the distinction between damp fuse and wet fuse; the former indicates cases in which a seemingly inconsiderable percentage of moisture, in either the air or stone, affected the burning of the fuse. During 1928, we greatly reduced the number of misfires due to this cause by procuring a brand of fuse more impervious to moisture.

Misfires: Treatment on Occurrence

In the preparation of these comments it was originally intended to submit an exhibit showing the various methods employed in treating misfires and the number of holes handled by each method during the 5-year period. It was found, however, that such an analysis would be of little value without full knowledge of all of the circumstances attendant in each case. The study did disclose, however, that at least 70 per cent. of all misfires were handled without disturbing either the explosive or the stemming in the misfired holes. With electric blasting caps, it was possible in a large number of the shots to fire the holes that originally

failed, by correcting defective wiring. Most of the cap-and-fuse misfires were in block holes, so that as a rule it was possible to shatter the block and detonate the explosive by inserting a new primer on top of the stemming, or by using a mud cap. When the fuse was cut by a flying spall, the fuse was relighted if it was long enough for safety; if not, a new primer or mud cap was used.

In nearly every instance where it was not possible to handle the various types of misfires by the methods described, part of the stemming was removed by using compressed air, a copper auger or a wooden or copper spoon. A new primer was then inserted and fired. In underground mining, as well as in quarrying, it is our practice to insert a known amount of stemming in the holes in all primary shots. This information is of value if removal of part of the stemming is required, but we have practically eliminated this necessity in underground mining by stemming only 16 to 24 in. instead of to the collar of the hole. As a rule, a new primer inserted on top of this amount of stemming will fire the original charge.

To the best of our knowledge, in the operation of 21 limestone mines and quarries and in the production of about 100,000,000 tons of stone over a period of more than 35 years, we have not experienced either a fatal or a lost-time accident in the attempt to remedy a misfire. We know this, certainly, to be true the last 10 years, in which approximately 34,000,000 tons of stone has been produced.

Improvements in Practice

In the light of our experience, various changes have been made in our practice and additional precautions have been taken, all leading toward better performance in the minimizing of the number of misfires as follows:

1. In one of the quarries, the average number of holes fired in each blast varies from 50 to 200 but sometimes reaches as many as 400. Better results have been obtained by firing with 220-volt alternating current, with the holes connected in parallel series, than with a blasting machine and a smaller number of holes in each shot. At that quarry during 1928, there was only one misfire in 693 blasts in which 48,000 holes were fired; that one was due to defective wiring.

2. In each of our underground mines, 300 to 500 holes are fired by electric blasting caps each day. If blasting machines are used, great care is exercised to avoid overloading the machine. In keeping the permanent lead wires advanced toward the working face as far as possible, the liability of defects of workmanship in the temporary wiring system is decreased.

3. In wet conditions, if ammonia dynamites are used, a gelatin primer will assist in assuring detonation. With those conditions it is often advisable to use electric blasting caps instead of cap and fuse. In each of the three quarries where the stone is hand-broken, 500 to 700 block-hole

shots are fired per day, and even though these are divided into four different firing periods, it is obviously impracticable to use electric blasting caps.

4. Care in the loading of the primer should be insisted upon. It is particularly necessary in the loading of block holes to prevent the tamping of a primer in which the cap is inserted in perhaps only a portion of a stick of dynamite. In primary holes, when practicable, a second cartridge should be used as a cushion between the primer and the tamping stick. We have found it advantageous to use a rubber tip, made from old belting and attached with wooden pegs to the tamping stick, to prevent brooming or wear.

5. The loop method of making the primer is, perhaps, the most desirable one. We have found that it can with care and some difficulty be used with 1½-in. gelatin dynamite, but with the more granular ammonia dynamites its use is less feasible, as the cartridge becomes torn and shapeless through punching the holes. With the latter dynamites, we are using the half-hitch method with the necessary care to avoid skinning the insulation on the leg wires.

6. We have never experienced a premature explosion due to stray currents. Nevertheless, in the underground mines, the precaution is taken to short-circuit the lead wires at the firing station and to test them with a squib before making the final connections at the face. The only similar case occurred in 1926 in one mine where one hole, of 19 loaded and connected in series, misfired during a severe thunderstorm. No lead wires had been connected to this series. Fortunately, at the approach of the storm the men had been notified to vacate all rooms in which holes had been loaded.

7. Both the operator and the manufacturer have a serious responsibility in making the proper selection of the type of explosive to be used for a specified duty and in making certain of its quality upon receipt, as well as at shipment. Uniformity in this quality is particularly essential. We believe that most manufacturers realize and have accepted this responsibility, and in meeting it they have succeeded to an extraordinary degree; but this fact should not act to reduce the rigidity of the inspection and testing by the operator.

8. After the receipt of explosives it is important that the operator exercise the greatest care to maintain a definite schedule of their use, so as to consume the oldest stock as soon as possible. In addition to the inspection of material as received, periodic tests are made to determine the average burning speed of fuse; all electric blasting caps are tested with a galvanometer before priming, a second time after they are connected in their circuit and a third time after the lead wires are connected. The operator should rightfully insist that the resistance of the caps of any one manufacturer be uniform within very small limits. By means of his first galvanometer test the operator is able to divide these caps further

into groups of more nearly equal resistance if he so desires, but this should not be necessary.

9. In each of our plants, in addition to the usual safety committees and meetings of foremen, a blasters' school in charge of the superintendent is maintained. He outlines, in monthly sessions, a definite course of instruction and study and discusses with the blaster foreman and his subordinates such current practices as require commendation or criticism. In maintaining the interest in these schools and their value to the employees, we have had the advantage of addresses given by technical representatives of the manufacturers of explosives and by representatives of the Department of Labor and Industry of the State of Pennsylvania, which has supervision over the operation of quarries and of mines other than coal mines. This educational work has been of great value.

10. No effort has been made to cover the multitude of causes leading to misfires and we have merely pointed out those precautions which have been found to be most often overlooked or considered as non-essential and costly.

Recommended Practice

Our experience as outlined would indicate the following recommendations in connection with misfires:

1. Select an intelligent and careful type of workman. Give him the advantage of competent supervision and of education in the technique of his tasks.

2. Use explosives and accessories designed for and best suited to the duty to be imposed upon them.

3. Establish a rigid and definite schedule of inspection of all explosives and accessories, both on their receipt and during their life.

4. Require practical perfection in the fulfilment of the manufacturer's obligations and responsibilities in furnishing these supplies; after their receipt, be equally particular in avoiding deterioration due to improper storage, age and misuse.

5. Maintain a record of all shots in as complete a form as is practicable in order to know where improvement in practice is required.

6. Do not issue rigid rules to cover the method of treating misfires on occurrence. The blaster foreman or other person in charge of the shooting should be thoroughly informed, by study and training, as to the various methods that have been found expedient to use in handling misfires. He should know, as well, the dangers that are most likely to be met in the practice of each of these methods under varying conditions. Misfires should be handled under the personal supervision of this man and then only after he has familiarized himself with the particular problem in hand. If in doubt as to the safest method to pursue, he should request the approval of his superiors. In the event of a difficult or partic-

ularly dangerous situation it may even be necessary to procure the assistance of technical advisors.

7. In the making of rules, be convinced that one is actually required. Make certain that it reaches the real source of trouble and is entirely applicable to all employees and conditions likely to come within its scope. Use at least as much care and intelligence in formulating the rule as is expected of the employee in its observance.

8. The fundamental facts governing the proper use of explosives are available to anyone interested in obtaining them. No doubt additional refinements and safeguards will be developed, yet there exists not so much the need for newer and more stringent rules as for the clarification of and greater familiarity with these basic principles. The ultimate responsibility for reducing to a minimum the hazards incident to the use of explosives lies almost entirely with the operator and his employees, and their success in discharging that responsibility will be measured by their intelligence, their knowledge and their cooperation.

DISCUSSION

(This discussion refers to the three preceding papers on misfires.)

T. D. Thomas.—A statement was made some time ago, and was repeated to me some weeks ago, that when *X* caps and *Y* caps and some other caps were put into water and put under 50 lb. pressure, the *X* caps and *Y* caps failed but the tetryl caps did not. It seemed a little peculiar to me. We tried the experiment and fired all three types. We made about 40 different tests on them. The tetryl caps were not any better than the other caps.

The idea of the tetryl cap, as stated in my report, was that it would prevent missed holes when using insensitive powder. Many coal companies receive complaints from customers that caps are found in their coal. The present type of fulminate cap if detonated will cause personal injury. To use a more powerful cap will produce a corresponding bodily injury. Occasionally complaints are received that caps explode in stoves and furnaces; in such instances, the more powerful the cap, the greater the damage. I do not believe in increasing this hazard.

S. P. Howell, Pittsburgh, Pa.—Mr. Thomas' point, I take it, is that the cure is to choose an explosive that is not quite so insensitive in preference to using a presumedly stronger detonator.

T. Marvin, Wilmington, Del.—In instances of misfires which have occurred because of insensitive explosive, Mr. Worthington, have you checked up in any instance, or in several, to find out whether the explosive was insensitive because of storage or whether the explosive was fresh?

A. W. Worthington.—There are few of those, if you refer to causes, shown on my Table 2 as "insensitive dynamite or weak caps." They are listed together. There is no way of telling, if the cap fires and the dynamite does not, which is at fault.

Answering your question directly—in instances where the misfire might have been caused by insensitive dynamite, I do not know definitely in every instance whether it had recently been furnished by the manufacturer or had deteriorated while we had it. Realizing our own responsibility in that matter, we have tried at all times to avoid any chance of using old dynamite, so that there is but little possibility of that

having caused the misfires. It is a mistake to try to figure out ways and means of using explosives that are not fit to be used.

S. P. HOWELL.—When a cap goes off and does not fire the explosive, the cap is gone. I know of instances in which a metal-mine official recovered the explosive from the bore-hole. They sent it to us at Pittsburgh. It was a gelatin dynamite. With it they sent some gelatin dynamite that they knew to be fresh. They wanted comparative tests. These explosives were in rather small quantities. Some samples were about one-half cartridge, or perhaps ¼ lb. We could not, of course, make the usual half-cartridge gap test, but we took a thin slice of the gelatin dynamite and placed it on the square end of a short oak stick.

We took blasting caps and pointed them toward this explosive, so that when the cap went off, the bottom of the cap would be projected bullet-like. Much to our surprise we found that the cap would fire this dynamite even though samples were as much as 3 ft. away from it.

We did, however, find a difference between the fresh gelatin dynamite and the dynamite from the hole. The fresh dynamite was substantially more sensitive. This method may be utilized to determine whether or not the explosive recovered from the hole is or is not of adequate sensitiveness and thereby give information to admit deductions as to whether it may be the detonator or explosive that is at fault.

C. S. HURTER, Wilmington, Del.—In regard to Mr. Thomas' statements about the bridge wires of exactly the same resistance, it is a physical impossibility to have bridge wires of identically the same resistance, but it is possible to weed them out by a close testing within resistance limits, so that if the proper amount of current is properly applied, they all should go. The application of the current is important. The current should not be allowed to build up from zero in the blasting circuit. That is the reason, for instance, that in the magnets on blasting machines they short-circuit the current during the stroke of the rack bar, so that the current is built up to full intensity by the time the rack bar reaches the end of its stroke and releases the current into the blasting circuit.

We have had series connections with 25-cycle current where misfires were apparently caused by the building-up effect between these cycles. For a long time we were under the impression that the minimum firing time between the application of a current and the explosion of an electric cap was 0.014 sec. Within the last year or so we have had much more elaborate experiments made and have found that with the more intense current the period of firing time is smaller. In fact, theoretically, if a 60-cycle current were used, there should not be more than 1½ amp. going through a series circuit.

We made experiments with 20 caps in a series and very heavy current, but we got no misfires with the 60-cycle current, which would indicate that in the shorter period of the half cycle of a 60-cycle current, the lag in heating is sufficient to overcome any difficulty from the building-up effect of that source.

In regard to what Mr. Worthington said, the conditions are so totally different between metal mining and non-metallic and coal mining, that one set of rules cannot be applicable to both. A number of metal mines that use water drills are successful in handling misfired shots. A jet is fixed on the water line, and with this jet and a stick of wood they work out the tamping and put in a new primer.

The thing to remember is that the most dangerous action on dynamite is a grinding friction; that is the most prolific source of trouble in handling misfires.

T. D. THOMAS.—What would you consider a fair variation in resistance of bridge wires?

C. S. HURTER.—I do not know our factor of resistance variation.

MEMBER.—I understand that our factor of resistance is to within ¼ ohm now. The old used to be ½ ohm.

T. D. THOMAS.—I have a series of tests on that. I think I found that running from 2.06 up to 2.38 ohms.

C. S. HURTER.—How long ago?

T. D. THOMAS.—In 1919. I think I have about 75 tests.

C. S. HURTER.—The change has been made comparatively recently.

T. D. THOMAS.—There were 30-ft. leg wires, No. 6 detonator. In that series we had 150 holes put in and 75 missed.

S. P. HOWELL.—I have a memorandum regarding *Report of Investigation* No. 2384 by the U. S. Bureau of Mines, "Failure of Center Holes in Blasting." Were those that failed in the center?

T. D. THOMAS.—No, they were not all in the center.

C. S. HURTER.—Where it is practical to use it, there will be fewer misfires with parallel connections than with series connections where it is possible to use a power current.

T. D. THOMAS.—This happened on our stripping work. In our mines we fire series Y-10 blasting cap.

C. S. HURTER.—It is extremely difficult, in some instances, to use parallel connections in underground work; Mr. Worthington's mine is an example. The tunnel people are going very largely to parallel connections with the power current in place of the series with or without the blasting machine.

T. D. THOMAS.—How many holes are they firing in those you speak of?

C. S. HURTER.—From 18 to 36.

A. W. WORTHINGTON.—We had that same trouble, Mr. Thomas, some years ago, with caps of varying resistances. In later years the process of manufacturing caps has been much improved. We have confirmed that by testing the caps before using them. As I remember it, the total normal resistance of our caps and leg wires is 1½ ohms. We get caps varying about ¼ ohm from this resistance. As a matter of fact, we would like to have them closer than that. One superintendent is so insistant on uniform resistances that he makes a subdivision of the caps into groups, with less variation. He gets them down to half that tolerance—that is, 0.12 ohm either side of the normal.

W. R. CHEDSEY, State College, Pa.—I realize that cushioned blasting in coal mines has not been increasing rapidly, as we thought it would, but I wonder if anybody can give us his experience as to the relative number of misfires in cushion blasting as compared to the non-cushion or tightly packed explosives.

S. P. HOWELL.—By cushion blasting, do you mean air space in front of the charge or loose stemming in front of the charge or air space around the charge?

W. R. CHEDSEY.—Most of the state laws prohibit an air space between the tamping and powder. Consequently the only alternative under the Pennyslvania law is

an air space between the bottom of the hole and the powder, or around the powder if the hole is of larger diameter than the stick.

S. P. Howell.—I cannot say regarding the air space at the bottom of the hole, but it is common in firing permissible explosives in bituminous coal mines to have an air space around the cartridge, and certainly misfires are not frequent when that method is used.

R. V. Ageton, Miami, Okla.—Our main trouble in the Tri-State zinc and lead district was not so much misfires as it was premature explosions. Prior to Mr. Howell's trip to the district two years ago, we had on the average of three or four deaths a year from premature explosions. Mr. Howell spent from four to six weeks

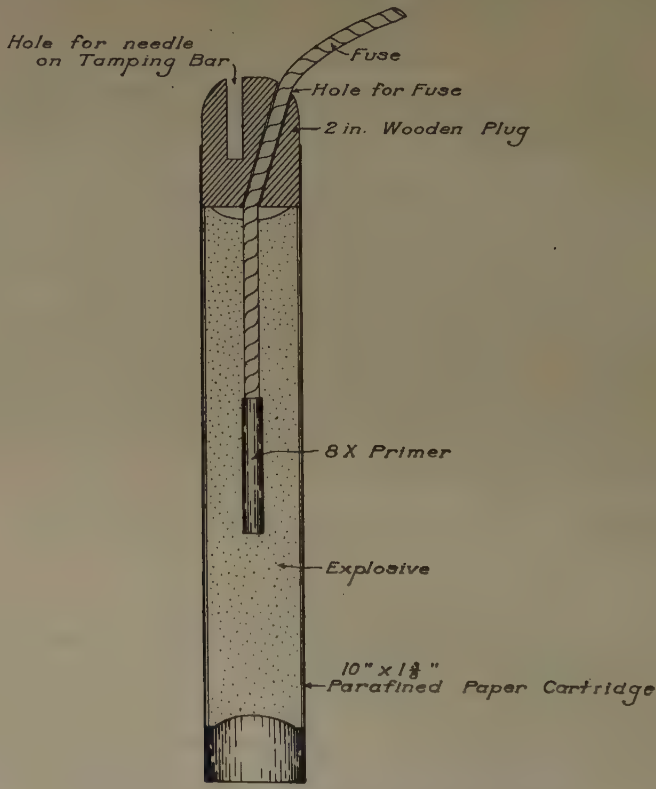


Fig. 3.—McLaughlin-Anderson shot tube.

with us. We had about 16 safety engineers working for the various companies and he went around with them and studied our whole situation. He made a complete report on his findings, with recommendations. We adopted the majority of the latter and in the past two years we have had no deaths caused by premature explosions.

We use one preventive measure which may be new to you. Two men down there, both of them practical mining men, one the superintendent of a property and the other the foreman, developed what they call the McLaughlin-Anderson shot tube (Fig. 3). I cannot tell you this is a panacea for premature explosions; all I can say is that we have never had a premature explosion when using this tube.

This shot tube is a cylindrical tube of paraffined paper, or some other waterproofed paper, about 10 in. long and 1⅜ in. dia. (Fig. 3). A wooden plug is inserted in one end of the tube. This plug has two holes, one for the fuse and one for the copper needle on the tamping bar. To place the powder in the shot tube, first thread the fuse through the hole in the wooden plug, then cap the fuse and put the capped fuse in the powder and the powder in the tube. Generally the fuse is threaded through the plug and the fuse is capped on the surface, all are then taken underground, where the powder is placed in the tubes as the "shots" are required. In this manner the caps, or primers, are protected from the time they leave the cap box until they are fired; furthermore, each cap is checked, so that there are few caps found in our muck piles.

The amount of powder put in a hole depends on the depth and the burden of rock on the hole. In drifting, raising and sinking, from 7 to 15 sticks of powder are used to the hole.

When shooting stope holes we "spring" them before blasting. Sometimes 100 lb. of powder is used in the stope hole after it has been chambered or sprung. We put in the stope holes about one-third of the charge, then the shot tube containing the detonator, then the rest of the powder, and then stem or tamp the hole with clay or fine dust.

In drift holes we put in about three cartridges of powder, then the shot tube, and more powder on top. Since Mr. Howell's visit, we have been using stemming or tamping in drift holes, something that has never been done before. We have been using these tubes in some of the mines for about three years and the cost of the tubes is very little.

There is another type of shot tube, a straight paper shell, but this type does not have some of the advantages of the McLaughlin-Anderson tube. I believe this McLaughlin-Anderson tube has a dual advantage—it not only keeps the primer from coming in contact with the walls of the hole, because of the paper shell, but the copper ferrule on the tamping bar fits to the hole in the wooden plug and thus the "shot" can be guided past fissures into the bottom of the hole without danger of the ferrule coming into contact with the primer.

We have had premature explosions with copper ferrules, copper shoes and copper pins. There is a question I wanted to ask after a while, regarding that. We are trying to keep all metal away from our powder and primers. The wooden plug does that. With the straight paper shell, the tamping bar goes into the powder, as before, and has another additional bad feature that it might cause a crimp in the fuse, which would not be likely with the McLaughlin-Anderson shot tube.

All of these shots, as they call them down there, these tubes, with the fuse and primer in them, are made up on the surface and kept underground. The shots are made up in one day's supply at a time. The tubes are purchased in 10,000 lots. In the last two years we have practically done away with the old system of every miner shooting his own round of holes. We have schools for the blasters, as we call them—not shot-firers. We get lecturers from powder companies to speak to them.

I want to emphasize one of Mr. Worthington's remarks. We have been doing this work for five years. We started out in 1923 with 29.7 days lost, through accidents, per 1000 shifts worked. In the first six months of 1928 we only lost 8.9 days due to accidents per 1000 days worked. Both figures are exclusive of fatalities. We have not used a rule book. We have no rules whatever, excepting those developed in our safety engineers' meeting, where we decide that certain things are good and from which the engineers for the various companies go out and talk to their foremen and ground bosses about what we have decided.

The reason for our lack of rules is just as Mr. Worthington said. Too often when one thing goes wrong, you think you have found the cure and make a rule. You may

cure only one-tenth of the trouble by that rule. In the last analysis, not only in accident prevention but in nearly any action of living, education is 99.99 per cent. of it.

C. S. HURTER.—With this arrangement of the tube, you would have to place a primer at the top of the charge, wouldn't you?

R. V. AGETON.—Not necessarily.

C. S. HURTER.—Have you had much difficulty with cut-off holes in drifting?

R. V. AGETON.—Some men think they get better breakage at the top, some at the bottom and some in the center. That is, regardless of all information on the subject, men think as they will, and with that tube you can put the shot anywhere you wish.

S. P. HOWELL.—The safety organization in the Tri-State zinc and lead ore producing district has been efficient during the last five years and anything I have done there was as a representative of the United States Bureau of Mines. We are always willing to cooperate to the fullest extent with all agencies having to do with accident prevention and mining.

This type of shot tube is particularly adapted to ragged holes or holes that have crevices opening into them. The difficulty in not using it arises from the fact that a primer of explosives is likely to become stuck in these crevices and cracks, and this has resulted in premature explosions. By reason of the rigidity given by these tubes, a special hole for the copper prong on one of them, and in the other the fact that this prong is held firmly between the stick of the explosive and the edge of the tube, permits those who put the primer in to "lead" it, as they say, into position, without losing it in any of these by-ways. It is not, I think, proposed for use in holes that are smooth or good, as holes are in most types of mining.

A. W. WORTHINGTON.—I gathered that your prematures were due to the fact that there were copper tips on the tamping bars. Was that necessary? Could you not use straight wood without metal tips?

R. V. AGETON.—We tried to use a hardwood needle on the end of the tamping bar, but as Mr. Howell explained, we are troubled by little fissures. They may be anywhere from ½ in. to ½ mile in diameter. If powder, the primer, or anything else gets into those holes the charge is lost, so when the powder is being loaded into a 30-ft. hole which has been chambered to hold two boxes of powder, each stick of powder must be threaded on the tamping bar on the needle, put back in its place, and released with a twist of the wrist; then the primer must be put in.

A. W. WORTHINGTON.—We do the same thing but we use a wooden plug that acts in the same way as the metal one.

R. V. AGETON.—In our particular instance we have a hard flint, which will wear out a hardwood needle, or whatever you want to call it, in a great deal less time than it will the other.

C. S. HURTER.—The point you bring out is that you use it in placing the powder, not in tamping it.

R. V. AGETON.—We do not tamp with the metal. Has anybody made any tests on the differential between different geological formations, such as limestone and flint, and between shale and coal, to find whether there are stray currents sufficient to set off charges? We have had some very weird premature explosions that we could not explain at all.

C. S. Hurter.—There have been a number of accidents in the Lake Superior Iron Co. district from stray currents, and I think there were some in the Scranton district in the D. & H.

S. P. Howell.—There are some records of alleged stray currents in the bituminous district. However, it is current that sets off the electric detonator and a simple measure of voltage is hardly a fair criterion. I suggest that where stray currents are suspected or known to exist, proper test would be to connect the electric detonator, or preferably an electric squib because of its greater safety. I have received the impression from Mr. Thomas that they actually did that in some cases. That proves beyond the shadow of a doubt that there is current adequate to set off the electric detonator.

A. W. Worthington.—Several times mention is made of state laws. Those governing coal mines have been in effect for a great many years. Now that other types of underground non-metallic mines are becoming more common, the states are taking cognizance of that fact and some of them are passing regulations. It is important that it be recognized that they are quite different and that metal mining differs from both.

Mr. Forbes mentioned pressing the entire charge into place at one time. Perhaps that can be done in a coal mine, but not in a limestone quarry or mine. Putting the cap in the bottom of the charge has been advocated by Mr. Forbes. Our objection to that in limestone mining is that there is more likelihood of damaging the leg wires or fuse. Also, putting it in the top seems to us a needless hazard, for if it is necessary to remove part of the stemming in order to fire a misfire, there is a needless disadvantage in having the cap in the top cartridge. That, in part, has led to the rule, "Never remove stemming," which is not at all the source of the trouble.

A number of articles have been written as to the advantage of having a cap pointed toward the bulk of the charge. We can realize the advantage of having the cap parallel with the long axis of any one particular cartridge, and not pointing across it, but we have not been able to find that it makes any difference whatever in having the cap in the top cartridge as compared to having it in the third or fourth cartridge from the top. There may be 2 ft. of explosive on top of the primer and 4 ft. below it, with the cartridge pointing toward the greater bulk.

Mr. Forbes and Mr. Thomas have as their standard method of handling misfires, in coal mining, the drilling of a second hole near the misfired one. As pointed out in our paper, we do not want to have that method set up as being required in limestone mining or quarrying, as the problems are entirely different.

T. D. Thomas.—In reference to stray current, we had, as I state in my report, found a difference in potential between the coal and the rock when we had all the switches in the mineral pulled out. Right above that section of our country there is a series of high-tension wires, and I have often wondered whether they might not have some leaky points. The big power plant is probably a mile away.

S. P. Howell.—That was one of the alleged reasons for stray current in a certain bituminous mine in Western Pennsylvania.

In reference to Mr. Worthington's remarks regarding the position of the primer in the hole, no doubt Mr. Forbes will have a chance to explain why they use it. In bituminous coal mines, permissible explosives are used, and they may be a little on the insensitive side. Also, they are usually air-spaced around the cartridge and are used in very small quantities, 1½ to 3 cartridges per charge, and are fired singly.

Member.—I spend considerable time hunting for stray currents. We do not have much trouble finding them even in mines where there is no electrical

machinery. In one place we found a current several times. It was discovered later that when the hoisting engineer was hoisting there was more current than we thought. One leg of the connection was made to the pipe line and one to the rail. Wherever there is any current, sometimes there would certainly be enough to fire an electric detonator. There is a hazard about shooting from a trolley line that should not be overlooked. If there is any gas in the place, or the place makes gas, there is a chance of an arc from the end of the lead lines, because the wire cannot be disconnected from the trolley the instant the cap fires. I think there is no doubt that explosions have been started by having current on the lead lines at the time the coal was breaking and gas was being liberated from the fresh broken coal.

S. P. Howell.—That this is true, we know, I think, from a paper read before this Institute a few years ago by A. C. Watts, in which he shows photographs of the glowing legs of the electric detonator and blasting cable near the face of a room in a bituminous coal mine.[1]

J. H. Smith, Wilmington, Del.—I feel it is advisable to inject a note of warning about the use of the Tri-State primer shell. It can be used with safety in that district, as the records have proved, but trouble and possibly accidents are likely to occur with its use in other districts where the method of blasting is different. The nature of the formations mined in the Tri-State are such that highly concentrated charges are necessary and, to bring this about, the bore-holes are chambered. Explosives which detonate at high velocity and which, therefore, are highly sensitive, are the most effective. In other districts where the rock is less brittle, slower acting explosives of much less sensitiveness, and used in unchambered bore-holes, are more effective. In using these slower acting explosives, especially of the permissible type, in unchambered bore-holes incomplete detonations are likely to occur, because of the presence of the wooden block in this primer shell. The primer as applied in the Tri-State is completely surrounded by the sensitive dynamite and the wooden block does not operate against complete detonation, but it can do this in an unchambered bore-hole when explosives of low sensitivity are used, especially if the primer is inserted into the bore-hole in a reverse position.

A number of years ago I talked with a man in the Tri-State district who had lost a hand and a shin bone through a premature explosion. In answer to my question as to the cause of the accident, he said that his coworker had side-primed the primer. In other words, the coworker had placed the cap at an angle in the cartridge so that the working end of the cap impinged against the side of the bore-hole when the primer was pushed home. This primer shell was devised to prevent this sort of thing.

C. S. Hurter.—I regret that I must take a little exception to that. Do you know the cartridge of powder and stick of wood system in shooting? I have done that successfully with 30 per cent. ammonia dynamite. I think the permissible runs about the same sensitiveness as a 30 per cent. ammonia.

J. H. Smith.—There will be a sensitiveness of 10 in. in a permissible powder whereas 30 per cent. will have a sensitiveness of 20 inches.

C. S. Hurter.—I have never seen it so large as 20 in. in 30 per cent. ammonia dynamite.

R. V. Ageton.—Let me emphasize the fact that I was merely telling you what we use; not recommending it for anyone else without thorough experimentation. I

---

[1] A. C. Watts: Experiments in Shot-firing with Low and High-voltage Currents. *Trans.* A. I. M. E. (1926) **74**, 512.

was merely calling your attention to a novel development in the Tri-State district, making no recommendations at all.

S. P. HOWELL.—Speaking as chairman of this Sub-Committee on the Use of Explosives, I want to say it is likely that the use of these shot tubes with permissible explosives would be quite improper because in adding it to the permissible explosive, you have changed the composition of the material you put into the bore-hole. The Bureau of Mines considers that the wrapper functions as an unmixed ingredient and has an effect not only on the temperature of explosion but also on the nature of the gaseous products of explosion, and might actually make the explosive non-permissible.

J. H. SMITH.—The permissible type of explosive is supplanting gelatins to a great extent in limestone mining.

S. P. HOWELL.—I presume so, but not because of their unique characteristic of having a flame of low temperature and lower duration; it is because of their unique strength characteristics.

C. S. HURTER.—Mr. Smith refers to a series of explosives of the general type of the ammonia nitrate permissible explosive but stronger, designed for use where the low flame feature is not required.

T. D. THOMAS.—It is necessary and desirable to have printed rules. We had an accident some time ago, and in Pennsylvania we have compensation laws. We prohibited the use of fuse at one of our operations. There was one miner who had a brother working in a cap factory where they made detonators. The miner decided he was not going to use electric firing batteries. We had all the hardware stores discontinue the sale of squibs and caps, but he bought the fuse downtown. They sold them to the farmers. He took a detonator, an instantaneous detonator, and cut it off a little above the fulminate of mercury, as he had been doing right along. This day his knife was not sharp enough. The fuse would not go into the little portion above the fulminate. He took a nail and commenced to round it out and got up to "Mr. Mike." He lost four fingers.

We took that case up before the compensation board. They asked us if we had rules and regulations governing that. We said "Yes." They asked us if we had any rules governing the cutting of those exploders in half, so that a man would know what he was doing. We will evidently lose the case because we did not say, "A miner shall not cut an electric detonator in half and insert a fuse." There is a value in rules.

S. P. HOWELL.—May I ask, Mr. Thomas, does not the anthracite law of Pennsylvania oblige the manufacturer of explosives to outline these rules and some vice-president or president of the company to sign it, and require that these rules must be promulgated by a responsible official of the company as well?

T. D. THOMAS.—They tell us that those rules are valueless.

S. P. HOWELL.—I understand that a competent agency has said that the Legislature of the State of Pennyslvania does not have authority to delegate its powers to the manufacturer of explosives or the operator to be binding upon the citizens of the state. There is a very important legal aspect involved.

A. W. WORTHINGTON.—Of course, coal-mine people have been through that years and years ago. In Pennyslvania, there is legislation governing, in detail, the operation of coal mines. Three years ago, two sets of regulations were promulgated by the Department of Labor and Industry—one covering deep pits and open quarries and the other covering all mines other than coal mines. In cautioning against indis-

criminate rule making, I did not mean to infer that there must be no rules, but that one rule must not be supposed to be effective in every type of operation.

The other point as to rules was made in connection with the handling of misfires. There, I feel it is a mistake to prescribe a definite procedure,—whether the rule is made by the state or employer, on account of the great variation in conditions. One cannot even make a fixed rule for one type of operation.

In looking over our 1928 performance, there were 94 misfires, 7 plants, making 13 per plant, or 1 a month. It is not imposing a great task to put the responsibility for the investigation of each particular misfire of one a month, occurring during the firing of an average of 18,000 lb. of explosives, directly on the blaster foreman, and to not allow any particular misfire to be handled either by the man originally responsible for it, or by anyone except the person best qualified and ultimately responsible at that plant.

R. V. Ageton.—Regarding rules, all I wanted to say was that due to our peculiar conditions, and because the Producers' Assn. office was merely a cooperation of about 53 companies, not a single company under one manager, we did not get up a set of rules. However, there must be regulations and we are just attempting now to pass new laws regarding the separation of lead and zinc mine regulation from coal mine regulation. What I meant was merely that rules did not fit in with our particular work.

R. N. Hosler, Harrisburg, Pa.—It has been said that "rules are made in order to be violated and that exceptions might be taken." There is a psychological effect to a code of rules from an educational standpoint that cannot be overlooked. I believe, therefore, that it is a good thing to have rules. I think Mr. Ageton has rules in the last analysis, according to my interpretation of rules, even though not printed. The very fact that they use that inversely in his district would indicate it. They have had a wonderful effect according to his own testimony. I believe that we cannot have too many rules, if for no other reason than their educational advantage.

D. Harrington, Washington, D. C.—Of all the points brought out today, the one that appeals to me as having really the most benefit from the viewpoint of prevention of accidents in connection with explosives is the matter of having a blasters' school. I have been in and around mines for 30 years. I have yet to know a single mine where the persons doing the blasting in the mine have been given any sort of definite instruction. There, I think, is the keynote of many of the accidents that occur in connection with blasting in mines.

R. N. Hosler.—Mr. Worthington has one plant listed in Maryland. Is it a quarry?

A. W. Worthington.—It is a quarry, but not in operation. It is called "Keystone Limestone Co."

J. J. Rutledge, Baltimore, Md.—Mr. Worthington brought out the conflict between two different sorts of mining. We have just had some hearings in our state on regulations, supplementing our statute law. We have obtained some valuable information through some of the remarks made by Mr. Hosler and Mr. Thomas. Utah, Wisconsin and Maryland, and perhaps California, can supplement their statute laws with regulations without going to their legislatures. I recommend that to you as practically what Mr. Ageton has done out in the Tri-State region.

B. F. Tillson, Franklin, N. J.—In one mine, for several years instruction in loading holes and the sequence of firing and standard rounds has been given. The practice is to install in the mine rescue-training building a model of a heading, in which

iron or steel pipes represent the holes. Such pipes are mounted horizontally in a plank partition to represent the face of a drift, and are also vertically mounted to represent the holes in a raise. The men are instructed in the proper method of loading dummy cartridges in the holes and priming them, and also go through a demonstration as to their sequence of ignition with the fuses of proper length.

Until the preprint of Mr. Thomas' paper came to my attention, I had never heard of tetryl as a detonating agent. I would be interested in receiving more technical data as to the particular properties of any new detonating substance, the conditions under which it is desirable to use it, as to whether the wave of detonation could be transmitted a greater distance through inert materials, with possibly the increased power or rate of detonation of this new material, or whether, for instance, it permitted equal security from No. 6 exploder, as people were prone to desire with No. 8 (although my personal experience years ago in testing one against the other was that No. 8 did not have the advantage it was supposed to have in actual practice).

I do not desire to discredit any of the statistics in regard to misfires, but I recently had an experience which I would not have thought possible if I had not experienced it myself. I wonder whether we have not blamed misfires for accidents that were not primarily the cause of misfires, and I say that because of this experience: Two men were firing a round in a drift. The ground was secure as far as they knew in the drilling operation when they started to fire. They went back after firing the cut. They were too seriously injured for me to learn much from their own lips. One man said they had not used a bar in trimming the ground after they had gone back, and just as they were ready to light the holes, there was a fall of back which crushed both of them. If that fall of back had been delayed 30 seconds, we would probably have charged that serious accident to a misfire of some description. It occurred just before the holes in the next round had been ignited.

D. Harrington, (written discussion).—These papers brought attention to a considerable number of interesting facts and factors not only in connection with their exact subject matter but also in connection with blasting practice in general as applied to mines other than metal mines.

Mr. Worthington's paper performed a useful service in bringing out the fallacy of trying to force on one branch of the mining industry rules and regulations, even laws, taken from experience in another branch of the industry in which the underlying conditions may be wholly different. There is no question that to require the drilling of a parallel hole in case of misfire in quarrying would in many instances lead to disaster, although the parallel hole practice probably is reasonably safe in coal mining.

The detailed data presented by Mr. Worthington in tabulated form over a 5-year period (1924 to 1928, inclusive), as well as the separate data for 1928, contain information not only as to misfires but also as to other details of blasting that is seldom available in any kind of mine office—coal, metal or non-metallic. Certainly it is exceptional to have details on the firing of approximately 4,000,000 blasting holes; and that it pays to keep the records and take suitable action on data secured from a study of those records is shown by the fact that the lost-time accident rate of the limestone quarries involved was but one-fifth of the rate for all limestone quarries of the United States. It is also remarkable to learn that with fourteen hundred forty-six 300-day workers in these quarries in 1928, there were no accidents from explosives.

The information in Table 1[2] compiled by Mr. Worthington, showing that the number of holes shot per misfire in 1928, was but 4050 with electric detonators as against 15,380 with ordinary running fuse and detonators, makes those of us who are ardent advocates of electric blasting "sit up and take notice." It is surprising to learn that, according to the figures for 1924 to 1928, misfires are likely to occur nearly

[2] See p. 229.

twice as frequently with electrical blasting as with running fuse and detonators. The causes of misfires with electrical blasting appear to be chiefly defective wiring (which of course can be guarded against by carefulness on the part of the shooters) and defective electrical detonators, which are more difficult to guard against. A number of misfires were found to be due to defective electric blasting machines, and this cause seems to be sufficiently serious to demand some fundamental alteration in blasting machine design. With fuse and "caps," nearly two-thirds of the misfires were held due to dampness in some manner; in general this can be remedied by additional care in the storage of fuse and in the methods employed in the placing of the fuse and detonator in the blasting hole.

Mr. Worthington's discussion on the treatment of misfires and improvements made in blasting practice can be read with profit by all persons who are interested in the use of explosives for blasting in any kind of mine. His statement that there have been no accidents (fatal or lost-time) in attempting to remedy a misfire in 35 years' operation of 21 limestone mines and quarries with production of 100,000,000 tons of stone shows that the methods used can not be far wrong. His recommended practice for blasting is well worth the study of mining men. In particular every mining company operating in the United States, whether in coal, metal or non-metallic mining, should take an interest in the possible establishment and maintenance of a blasters' school such as described briefly by Mr. Worthington; the present-day practice in general is virtually to "pitchfork" the miner or shooter into his work and let him sink or swim as to what he does or does not do in the blasting-down of material mined. At any rate, there is on record no case, in so far as my information goes, where the mining company has actually taken the time and trouble to assemble its employees engaged in blasting to instruct them in safe, efficient methods of blasting; in a relatively few cases a few rules are formulated, but usually these rules are but skeletonized instructions which mean little or nothing in educating the user of explosives in correct blasting practice. If this symposium of papers on misfires brought forward nothing but the idea of the blasters' school, the papers would have far more than repaid all of the effort attached to their preparation, delivery and discussion.

The papers by Messrs. Forbes and Thomas on misfires in bituminous and anthracite blasting parallel each other in a number of essential features; both advocate the use of electrical blasting in coal mining and both give definitely good reasons for this advocacy; both recommend that in case of misfire if it is found impracticable to fire the missed hole, the best practice is to drill and fire a parallel hole. Mr. Forbes' 19 rules for handling explosives are in general very good, though some of us are decidedly opposed to blasting of any kind in any mine while the general working shift is in the mine. Mr. Forbes' "Conclusions" also are to the point, and in particular conclusion No. 7—"Maintain the same strict supervision in connection with the charging and firing of shots that is usually given to other phases of mining." In general there is so nearly no supervision over blasting that it can almost be said that blasting practice in mines is very much like Topsy in Uncle Tom's Cabin.

Mr. Thomas' paper on misfires in anthracite coal mines shows that at least some of the anthracite companies are alive to the necessity of trying to reduce the admittedly excessive accident rate from explosives in anthracite mining. The use of electric blasting is certainly a step in the right direction, and the details which Mr. Thomas supplies concerning difficulties in connection with electrical blasting show that electrical blasting has numerous weaknesses which must be guarded against if this relatively safe method of coal-mine blasting is to be kept reasonably safe and efficient.

It is indeed time to call attention to defects in the detonators as received from the manufacturer; to indicate the numerous methods by which the present-day blasting

machines fail in whole or in part; to show that blasting from the trolley wire is too dangerous to be tolerated; and to indicate the laxity of placing of wires for blasting, unless very careful supervision is kept over these installations. The giving of much of the surrounding detail in connection with accidents from misfires in electrical blasting is of much value to those who are using electrical blasting and may at any time encounter misfires or other troubles. The rule under which return to a misfire must await the following day, while drastic, is wise. The fact that explosives, and especially permissible explosives, when left in the mine rather quickly become insensitive and hence will be likely to cause misfires is something which cannot be stressed too much or too frequently.

There is no question that these papers have much "meat" in them; if it is at all feasible they should be circulated widely and should be given discussion in keeping with the wealth of information contained in them.

Hindered-settling Classification of Feed to Coal-washing Tables*

By B. M. Bird† and H. F. Yancey,‡ Seattle, Wash.

(New York Meeting, February, 1928)

During the past four years the experimental work in coal washing carried on by the U. S. Bureau of Mines and the University of Washington has been devoted mainly to the development of special methods for the tabling of fine sizes of coals that are difficult to wash. One important outgrowth of this investigation has been the application of hindered-settling classification to coal tabling.

Although used for many years in ore dressing, the term "classification" as used here may be new to many readers. It may be defined as the separation obtained when a bed of particles is held in a uniform state of suspension in an upward current of water, either continuous or pulsating, just sufficiently strong to keep the bed in a mobile condition. A common example of this type of separation is a jig in which there is no suction stroke. When the hindered-settling classifier is used with the coal-washing table, the fine-size raw coal, instead of being fed direct to tables, as is the prevailing practice, is first fed to the classifier and grouped, according to the rate of settling against the upward current of water, into perhaps six separate products. After dewatering, each of these except the sixth, a refuse product, is elevated to a seperate feed bin and washed on a separate table.

This paper gives a brief discussion of the basis for the use of hindered-settling classification of a table feed; a description of the experimental work; a comparison of the results obtained with and without classification; and a discussion of the application of the classifier in commercial coal washing.

Basis for Use of Hindered-settling Classification of Table Feed

Although theoretical reasoning might be used to justify treating a table feed by hindered-settling classification,[1] the simplest method of demonstrating the need of such a step before tabling is to show the

* Published by the permission of the Director, U. S. Bureau of Mines.

† Supervising Engineer, U. S. Bureau of Mines.

‡ Associate Chemist, U. S. Bureau of Mines.

[1] A. W. Fahrenwald: The Theory of Stratification and Its Application in Ore-dressing. *Min. & Met.* (1926) **7**, 422; also many other papers.

separation effected in the table washing of a bony coal, as in Table 1.[2] These data resulted from submitting to float-and-sink and screen-sizing tests, 1 (–) ft. zonal samples from a table treating raw coal through 3/8 in. and on 20 mesh.

TABLE 1.—*Normal Sizing Action of a Coal-washing Table*

Percentage Weights Are in Terms of the Total Quantity of the Given Specific Gravity Found in the Feed. The Average Diameters of Particles Were Calculated from Screen Tests

Product	"Coal" under 1.38 Sp. Gr.		"Light Bone" 1.38 to 1.50 Sp. Gr.		"Heavy Bone" 1.50 to 1.70 Sp. Gr.		"Refuse" over 1.70 Sp. Gr.	
Per Cent. of Feed	57.3		17.4		10.7		14.6	
Zone No.	Wt. Per Cent.	Av. Dia., Mm.	Wt. Per Cent.	Av. Dia., Mm.	Wt. Per Cent.	Av. Dia., Mm.	Wt. Per Cent.	Av. Dia., Mm.
1	(*a*)		(*a*)		(*a*)		(*a*)	
2	(*a*)		(*a*)		(*a*)		(*a*)	
3	3.4	5.5	0.0		0.0		0.0	
4	4.9	5.4	0.0		0.0		0.0	
5	9.4	5.2	2.0	6.3	0.0		0.0	
6	10.5	4.6	4.2	6.2	0.0		0.0	
7	8.3	4.0	3.3	6.1	0.0		0.0	
8	9.4	3.9	6.5	5.6	0.0		0.0	
9	7.9	3.7	5.9	5.4	0.0		0.0	
10	6.4	3.6	6.0	5.3	0.0		0.0	
11	6.1	3.4	7.0	4.6	0.0		0.0	
12	7.0	3.3	8.4	4.1	3.0	5.9	0.0	
13	7.7	3.3	10.3	3.8	4.4	5.7	0.0	
14	4.6	3.2	6.7	3.6	2.8	5.7	0.0	
15	2.1	3.1	2.8	3.5	1.1	5.5	0.0	
16	2.4	3.1	2.5	3.6	1.6	5.3	0.0	
17	0.4	2.8	0.6	3.7	0.9	5.3	0.0	
18	1.8	2.9	2.8	3.6	2.0	5.3	0.0	
19	1.7	2.5	3.9	3.6	3.3	5.3	0.0	
20	5.0	1.8	24.1	3.2	70.9	3.7	23.2	5.2
21	1.0	1.3	3.0	1.6	10.0	2.7	60.0	4.3
22	0.0		0.0		0.0		16.8	3.2

(*a*) In this test only water was caught in these zones, 1 and 2.

If we examine first the separation according to specific gravity, we find "coal" in every zone except the last; the maximum percentage in any zone is 10.5, in zone 6, while 5.0 per cent. is mixed with a large proportion of "refuse" in zone 20. Only the separation of the "refuse" is good;

[2] B. M. Bird: The Sizing Action of a Coal-washing Table. U. S. Bur. Mines *Repts. of Invest.* Ser. 2755 (1926).

this is concentrated in the last three zones. If we turn now to the figures recording the average sizes of these various products, we find the cause of the poor separation of coal and bone. Clearly, in addition to a separation according to specific gravity, there is one according to size. If we follow down the figures showing the average diameters of the "coal" particles in each zone, we find a gradual decrease in size. For example, the "coal" particles in zone 21 are on the average less than one-fourth the diameter of those in zone 3. "Light bone" particles show this same gradation in size, decreasing from 6.3 mm. to 1.6 mm. along the table. This same separation according to size is apparent in the other specific gravity fractions. Evidently, then, separation according to size tends in general to throw coarse particles into the first zones and smaller particles into later zones, with little regard to specific gravity. Obviously if some means could be found to secure among the particles in the table feed a size-specific-gravity relationship such that the "coal" particles would be coarser than the "bone" and "refuse," this sizing-action, in place of opposing the separation according to specific gravity, would actually aid it.

In hindered-settling classification this identical size relationship of particles is secured, as may be seen by referring to Table 6, which contains the results of classifying a sample of coal passing a 407 Ton-Cap screen. For example, in compartments 1 and 2 the average diameters of the four specific gravity fractions, from lowest to highest, are as follows: 1.16, 0.73, 0.53 and 0.33 mm. respectively. Obviously, with a feed of this character the optimum conditions exist for a sharp separation on the table. The relative sizes of the particles, as well as their differences in specific gravity, favor a sharp separation.

The principles just discussed will aid in explaining facts developed throughout the remainder of the paper, which is devoted to the results of washing a carload of coal from the state of Colorado.

Experimental Work

The experiments on which this paper is based comprised in all 19 tests on a full-size table and 16 tests on a semicommercial unit of the classifier. In all of the tests the performance of the apparatus was checked by float-and-sink tests and sometimes by screen-sizing tests. Eight of the table tests were made on the same unsized feed but with varying adjustments and tonnages in order to get the very best possible results by that method. The two best of these tests, one at 4.9 and the other at 7.8 tons per hour, are given in this paper. With the exception of the product of the first compartment of the classifier, only one test was made on each of the classified products. Several different possible arrangements of the classifier and table were tried but only that showing the best results at 8.0 to 9.0 per cent. ash is given here.

Coal Used for Tests

The washing characteristics of the coal used in the various tests are shown very completely by the float-and-sink and screen-sizing data. However, it may be well to say that the relative proportions of the specific gravity fractions present and the relative sizes of particles in each determine in large measure the difficulty of the separation. From the standpoint of specific gravity any given coal is easy or difficult to wash depending on how large a proportion of it is composed of particles having specific gravities near to that at which the coal must be separated to give the required ash content in the washed coal. For example, if the separation is made at 1.50 specific gravity—that is, everything of less than 1.50 specific gravity is, so far as possible, to be included in the washed coal and everything of over 1.50 specific gravity to be removed in the refuse—the difficulty of the washing problem so far as specific gravity is concerned will depend mainly on how much of the feed is between the specific gravities of 1.40 and 1.60. From the standpoint of size, also, a coal may be classed as difficult or easy to wash by any given process according to the relative sizes of the coal and of the impurities—a factor to be discussed more completely later in the paper as it affects tables—and according to the proportion of sizes that are beyond the best working range of the given process. This Colorado coal may be classed as comparatively easy to wash on tables to any ash content down to 10 per cent., and as increasingly difficult to wash with each decrease in ash content below that figure, until at 8 per cent. ash it presents a very difficult washing problem.

The size of feed in all tests was the undersize of a 407 Ton-Cap screen, which gives a product like that of a standard Tyler 4-mesh sieve. The coal washed in an unsized condition was obtained by screening the undersize from the raw coal. The coal washed by classification and tabling was the same except that it was necessary, in order to have large enough classified products for tabling, to crush and include all of the oversize. This procedure increased the feed-ash of the classifier to 15.8 per cent. as compared to 15.0 per cent. in the table tests of unsized feed, and also increased the proportions of bone, thereby rendering the feed more difficult to wash than that treated in an unsized condition. Although it would have been an advantage to have some additional tests on this new feed in an unsized condition, to show more accurately the advantages of classification, it was felt that the additional expense was not justified, inasmuch as the results of the combination of classification and tabling were better in spite of the handicap and therefore demonstrated the relative merits of the two systems.

Apparatus Used in Experiments

The table used in these experiments was a full-size standard make, except that the riffles were higher and were spaced more closely than

those originally furnished with the deck. The riffles had a height of 1 in. at the head-motion, and gradually tapered toward the refuse end. They were spaced 2-in. centers except for the test of the finest of the classified products, where they were spaced 1 in. The advantages of this style of riffling over that ordinarily furnished with coal-washing tables have been described elsewhere;[3] it suffices to say here that it causes a distinct improvement both in sharpness of separation and in increased capacity of the table, especially in treating unsized feeds.

The classifier was a special six-compartment apparatus developed during this tabling investigation. Essentially it may be described as a sorting column in which the upper portions have been arranged so that each part of the column may be supplied with an upward velocity of water suited to the sizes of particles at that point. By this arrangement a maximum hindered-settling condition can be maintained in the classifier for materials of all settling rates; also, the amount of water used in treating each kind of material can be kept at a minimum. The products are numbered in the order of their settling rates, from lowest to highest.

Conduct of Experiments

In adjusting the table, an effort was made to secure good "distribution;"[4] that is, one in which the coal was fanned over the entire deck with only a small proportion—about 5 per cent. of the feed—discharged in the first 2 ft. along the part of that deck next to the head motion. The classifier was adjusted to maintain uniform hindered-settling conditions and to give the desired proportions from each compartment.

During the period of tuning-up either the classifier or the table, the washed products were collected in one compartment of the sludge tank, dewatered, and returned to the feed bin; thus a closed circuit was completed. The remainder of the test followed either of two procedures: (1) The entire sample, including the portion circulated, was washed as a "batch" test; the losses in slimes were determined by wasting all water through a standard orifice and taking samples to determine the percentage and character of solids; or, (2), the apparatus was fed with a fresh supply of coal, not before circulated, and, when operating smoothly a "time" sample consisting of a number of separate products including both coal and water was caught. In this latter instance all solids were precipitated from the water with hydrochloric acid, and were weighed with the remainder of the products. The classifier run described in this paper was a "batch" test; the table runs were "time" tests. All weights of products were put on a uniform dry basis and all analyses on a moisture-free basis.

---

[3] B. M. Bird: Coal-washing Research Will Save and Better Coal While Increasing Capacity. *Coal Age* (1927) **31**, 670.

[4] B. M. Bird: *Op. cit.*

Results of Experiments

Method of Interpretation

The best method of interpreting the results secured with coal-washing apparatus is that of comparison with those shown possible by correct float-and-sink tests of the feed. This is exemplified in Fig. 1, where the results of the two table tests (Nos. 3 and 5 from Table 2) on the same unsized feed are plotted in the form of curves.[5] One curve shows the float-and-sink test of the feed, representing the maximum possible recovery of

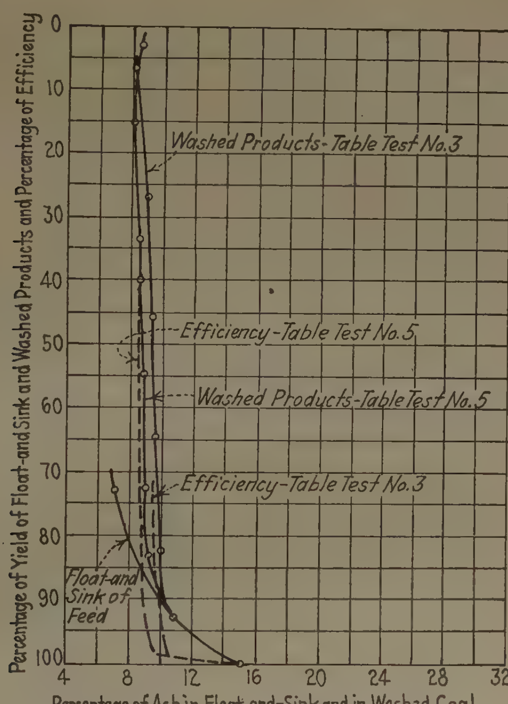


Fig. 1.—Table tests Nos. 3 and 5. Feed unsized through 407 Ton-Cap.

coal of each ash content. The two other curves in solid lines show the yields of washed products secured in the tests. The two curves in dotted lines show the true efficiencies of the separations at any given ash content; that is, the yields of washed products of that ash content divided by the per cent. shown to be in the original feed by float-and-sink.

In addition to its accuracy, which is, of course, the first consideration, this efficiency method has distinct advantages in interpreting the results of experimental work. Because the data, both of the feed and of the

---

[5] T. Fraser and H. F. Yancey: Interpretation of the Results of Coal-washing Tests. *Trans.* (1923) **69**, 459.

washed products, are reduced to curves, a washed product of any desired ash content may be selected irrespective of the ash contents of the products actually made during the tests; thus the available yield and the efficiency of washing are known at once. Also, by a simple arithmetical calculation, to be illustrated later, the proportions of middlings and refuse together with their respective ash contents can be determined from the curves of washed products.

TABLE 2.—*Table Tests Nos.* 3 *and* 5, *Feed Unsized through* 407 *Ton-Cap*

Product	Specific Gravity and Zones*	Weight, Per Cent.	Ash,† Per Cent.	Cumul. Wt., Per Cent.	Cumul.† Ash, Per Cent.
Float-and-sink of feed to table tests Nos. 5 and 3	Under 1.38	72.8	7.0	72.8	7.0
	1.38–1.50	13.3	20.2	86.1	9.0
	1.50–1.70	6.5	34.5	92.6	10.8
	Over 1.70	7.4	67.1	100.0	15.0
Washed products table test No. 5	1 and 2	2.8	8.5	2.8	8.5
	3 and 4	12.3	8.0	15.1	8.1
	5 and 6	18.3	8.9	33.4	8.5
	7 and 8	21.1	9.3	54.5	8.8
	9 and 10	18.0	9.3	72.5	8.9
	11 and 12	10.6	10.4	83.1	9.1
	13, 14 and 15	3.5	18.1	86.6	9.5
	16 and 17	1.6	24.9	88.2	9.8
	18 and 19	3.2	34.2	91.4	10.6
	20 and 21	8.6	61.3	100.0	15.0
Washed products table test No. 3	1 and 2	6.6	8.2	6.6	8.2
	3 and 4	20.6	9.3	27.2	9.0
	5 and 6	18.5	9.6	45.7	9.3
	7 and 8	18.7	10.2	64.4	9.5
	9 and 10	17.8	11.6	82.2	10.0
	11 and 12	8.1	13.7	90.3	10.3
	13 and 14	0.6	26.4	90.9	10.4
	15, 16 and 17	1.4	36.1	92.3	10.8
	18 and 19	0.5	39.6	92.8	11.0
	20, 21 and 22	7.2	66.6	100.0	15.0

* One-foot zones.
† Moisture-free basis.

It is, of course, axiomatic that if the comparison of two tests is to be exact the yield-ash curves of the feeds must be identical. But in the comparison given later, between table efficiencies with classified and with unsized feed, this factor has been ignored, inasmuch as it tends to favor the results with unsized feeds.

When this method of measuring efficiency of washing is applied to the results of several apparatus taken together, some rational means must

be found of combining the washed products. One's first thought would be to add the yields of washed coals of the same ash content from each apparatus. The result is exemplified in the following table taken from Figs. 2, 3, 4 and 5, for a washed coal of 9.5 per cent. ash:

Table test No.	19	11	12	17
Yield from each table test, per cent.	100.0	98.6	94.5	41.5
Instantaneous ash, or product of maximum ash content included in the washed coal, per cent.	80.0	44.0	24.6	10.7

Total yield of washed coal = 72.3 per cent.

This mode of combining results puts products with 80 per cent. ash from table test No. 19 with the total washed coal, and throws into

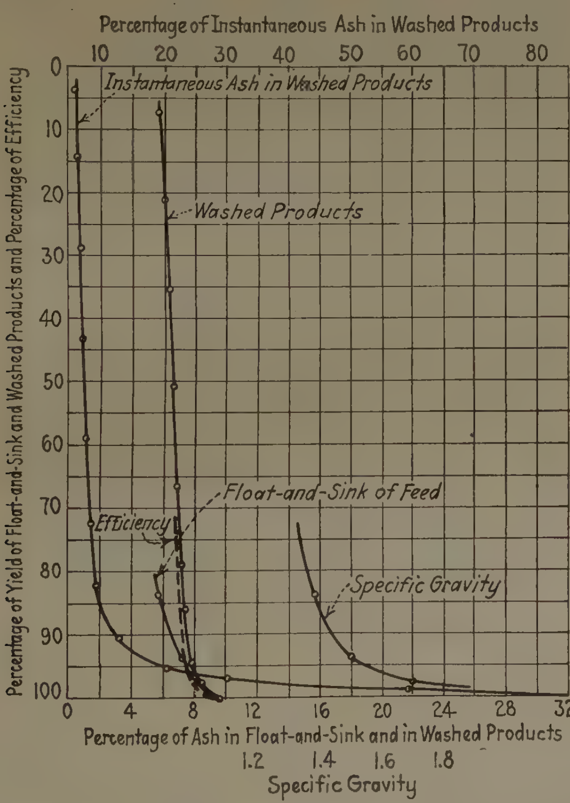


FIG. 2.—TABLE TEST NO. 19. FEED, COMPARTMENTS NOS. 1 AND 2 FROM CLASSIFIER.

middlings or refuse the products just above 10.7 per cent. ash from table test No. 17. Such a procedure would obviously show a yield much lower than that actually secured by the tables. However, these data, although

showing an incorrect method, indicate clearly that the products of the individual apparatus should be divided, not on a basis of the ash content of the washed coal, but on a basis of the product of maximum ash con-

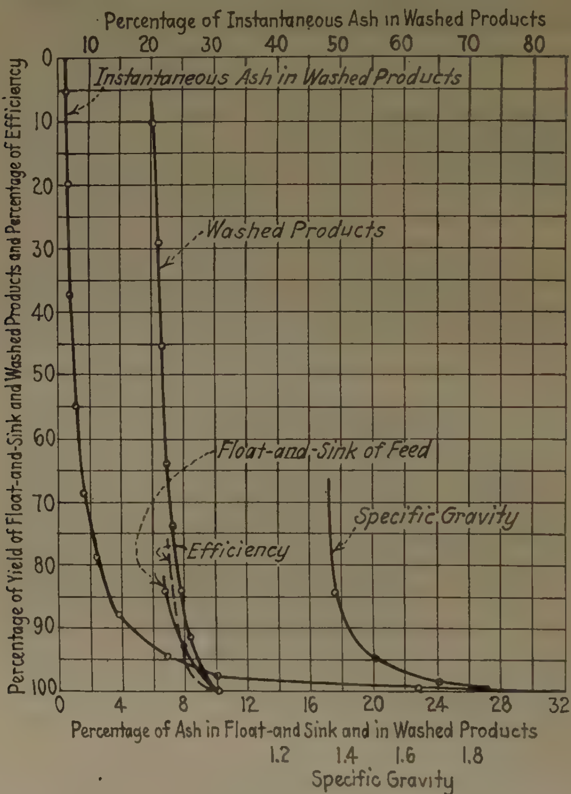


FIG. 3.—TABLE TEST NO. 11. FEED, COMPARTMENT NO. 3 FROM CLASSIFIER.

tent included in the washed coal. If these same data are re-combined on this basis for a washed coal of 9.5 per cent. ash, the "instantaneous ash" from each table is 22.0 per cent. and the total yield is increased from 72.3 to 79.9 per cent. as shown in the following table:

Table test No.	19	11	12	17
Yield from each table test, per cent.	95.5	94.3	93.2	87.2
Washed coal ash, per cent.	7.9	8.6	9.3	12.2
Sp. gr. at point separation	1.56	1.49	1.49	1.48

Total yield washed coal = 79.9 per cent.

If a series of yields of washed coals and of corresponding ash contents were determined in this same manner for different "instantaneous"

ashes, a curve could be drawn that would correctly represent at all points the combined results of all of the tests. The method, however, has not been used in this paper, because the "instantaneous" ash curves are only approximate. As accurate ones can be drawn only by resorting to calculus, the more direct procedure has been adopted of adding together in the order of increasing ash contents the individual products of the table tests shown in Table 3 and of combining with them three products

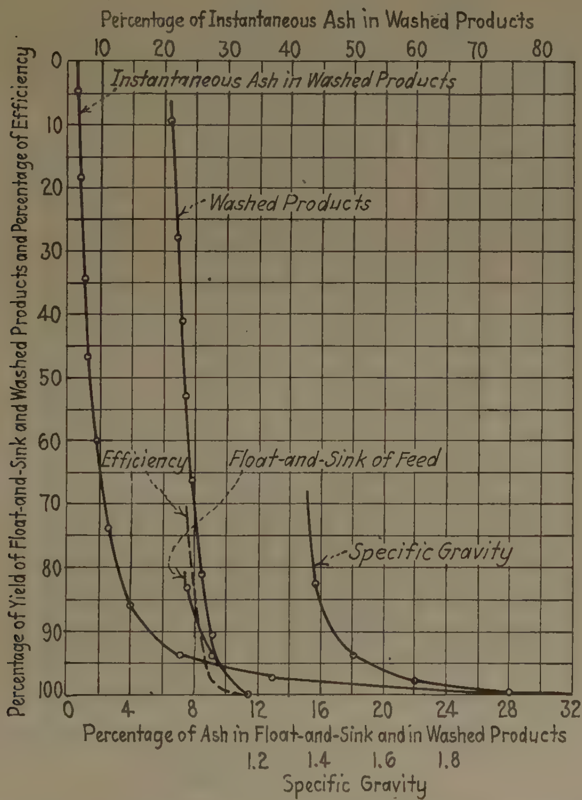


FIG. 4.—TABLE TEST No. 12. FEED, COMPARTMENT No. 4 FROM CLASSIFIER.

of the classifier which were not tabled. This procedure gives the data shown in Table 5 and in Fig. 6. Theoretically the individual products would need to be infinitely small to show a total yield as high as actually secured with the tables; but for practical purposes the addition of small products in this manner will give a result nearly enough like that obtainable in commercial washing to be entirely satisfactory.

As four washed coals of different ash contents, 7.9, 8.6, 9.3 and 12.2 per cent. respectively, are combined in the example just given to make

a washed coal of 9.5 per cent., it might appear that in commercial practice the location of the dividing point on the tables between washed coal and the remaining products would be difficult to determine, and that the results obtained by this combination could not be duplicated. But the specific gravities given with these data, which have been taken from the curves in Figs. 2, 3, 4 and 5, indicate that if the feeds to the individual tests are divided at approximately 1.49 sp. gr., the desired ash

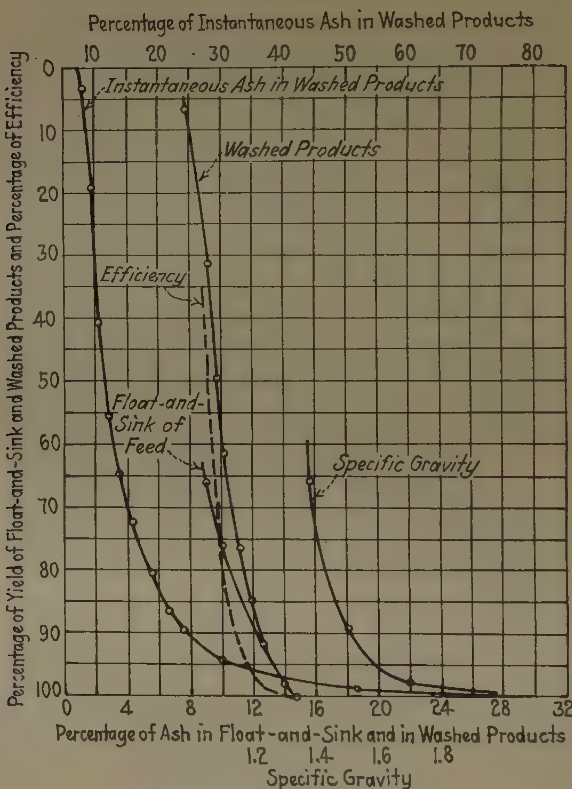


FIG. 5.—TABLE TEST NO. 17. FEED, COMPARTMENT NO. 5 FROM CLASSIFIER.

content will be secured in each of the washed coals. Evidently a solution of constant specific gravity, perhaps 1.45, could be used in making a washed coal of 9.5 per cent. ash, to control the division point between washed coal, and middlings or refuse products, on all tables. Likewise it is obvious that if a plant is using chemical methods of control, the "instantaneous ashes" would serve to control the separation equally as well as the ash content of the washed coal. In the instance of the separation at 9.5 per cent. ash, for example, the divider separating the washed coal from the

TABLE 3.—*Table Tests of Products of Classifier*

Product	Specific Gravity and Zones*	Weight, Per Cent.	Ash,† Per Cent.	Cumul. Wt., Per Cent.	Cumul.† Ash, Per Cent.
Table test No. 19					
Float-and-sink of feed: Comp. 1 and 2 of classifier test No. 14, 22.6 per cent. of classifier feed.	Under 1.38	83.5	5.7	83.5	5.7
	1.38–1.50	10.1	19.4	93.6	7.2
	1.50–1.70	4.0	33.1	97.6	8.2
	Over 1.70	2.4	62.0	100.0	9.5
Washed products................	1 and 2	7.2	5.8	7.2	5.8
	3 and 4	14.3	6.4	21.5	6.2
	5 and 6	14.0	6.9	35.5	6.5
	7 and 8	15.3	6.9	50.7	6.6
	9 and 10	16.0	7.7	66.7	6.9
	11 and 12	11.9	8.4	78.6	7.1
	13 and 14	7.4	9.6	86.0	7.3
	15 and 16	8.3	12.7	94.3	7.8
	17 and 18	1.6	20.5	95.9	8.0
	10 and 20	1.7	30.0	97.6	8.4
	21 and 22	2.3	58.9	100.0	9.5
Table test No. 11					
Float-and-sink of feed: Comp. 3 of classifier test No. 14, 20.1 per cent. of classifier feed.	Under 1.38	84.2	6.8	84.2	6.8
	1.38–1.50	10.7	20.2	94.9	8.3
	1.50–1.70	3.6	34.7	98.5	9.3
	Over 1.70	1.5	61.2	100.0	10.1
Washed products................	1 and 2	10.5	6.0	10.5	6.0
	3 and 4	18.6	6.6	29.1	6.4
	5 and 6	16.3	6.9	45.4	6.6
	7 and 8	18.5	7.7	63.9	6.9
	9 and 10	10.0	9.1	73.9	7.2
	11 and 12	10.2	11.0	84.1	7.7
	13 and 14	7.3	14.1	91.4	8.2
	15 and 16	5.4	22.1	96.8	9.0
	17 and 18	2.0	29.8	98.8	9.4
	19 and 20	0.9	62.0	99.7	9.8
	21 and 22	0.3	79.5	100.0	10.1
Table test No. 12					
Float-and-sink of feed: Comp. 4 of classifier test No. 14, 22.1 per cent. of classifier feed.	Under 1.38	82.8	7.6	82.8	7.6
	1.38–1.50	10.9	20.9	93.7	9.1
	1.50–1.70	4.2	35.3	97.9	10.3
	Over 1.70	2.1	62.8	100.0	11.4
Washed products................	1 and 2	9.0	6.4	9.0	6.4
	3 and 4	18.6	7.0	27.6	6.8
	5 and 6	12.9	7.6	40.5	7.1
	7 and 8	12.2	8.1	52.7	7.3
	9 and 10	13.4	9.4	66.1	7.7
	11 and 12	14.7	11.3	80.8	8.4
	13 and 14	9.7	14.6	90.5	9.0
	15 and 16	6.1	22.7	96.6	9.9
	17 and 18	1.0	37.0	97.6	10.2
	19 and 20	1.5	52.8	99.1	10.8
	21 and 22	0.9	74.9	100.0	11.4
Table test No. 17					
Float-and-sink of feed: Comp. 1 to 5 incl. of classifier test No. 16, 21.6 per cent. of feed to classifier test No. 14.	Under 1.38	65.7	9.0	65.7	9.0
	1.38–1.50	23.3	20.6	89.0	12.0
	1.50–1.70	9.0	32.7	98.0	13.9
	Over 1.70	2.0	54.3	100.0	14.7
Washed products................	1 and 2	6.5	7.8	6.5	7.8
	3 and 4	24.7	9.6	31.2	9.2
	5 and	18.6	10.5	49.8	9.7
	7 and 8	11.6	12.1	61.4	10.2
	9 and 10	6.7	13.4	68.1	10.5
	11 and 12	8.4	16.0	76.5	11.1
	13 and 14	8.6	19.0	85.1	11.9
	15 and 16	2.9	21.4	88.0	12.2
	17 and 18	3.1	23.6	91.1	12.6
	19 and 20	6.3	29.8	97.4	13.7
	21 and 22	2.6	51.0	100.0	14.7

* One foot zones.
† Moisture-free basis.

other products would be set at 22.0 per cent. ash. Thus the determination of the correct ash content of the washed coal to be made from each table,

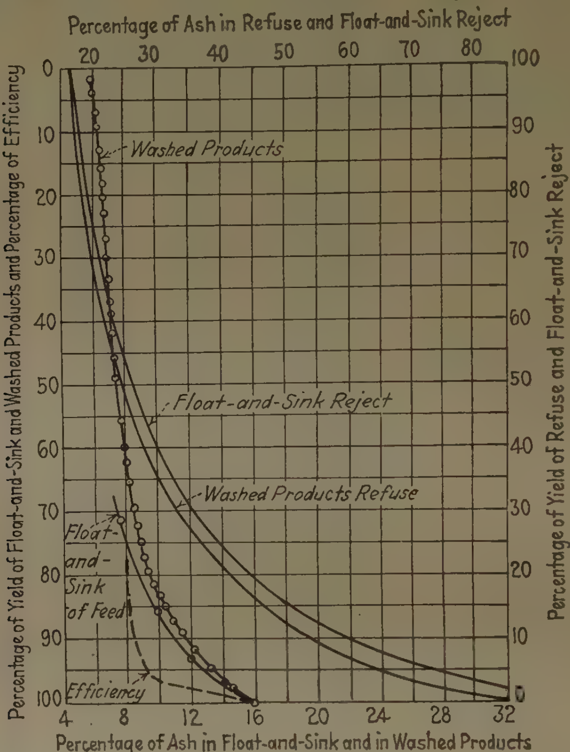


FIG. 6.—COMBINED RESULTS CLASSIFIER TESTS NOS. 14 AND 16 AND TABLE TESTS NOS. 19, 11, 12 AND 17.

a problem somewhat difficult of solution in the laboratory, will be simple in the commercial washery.

TABLE 4.—*Table Adjustments*

Table test	No. 3	No. 5	No. 11	No. 12	No. 17	No. 19
Tonnage washed per hr	7.8	4.9	11.3	11.4	12.0	6.6
Length of stroke (in.)	1.0	1.0	0.75	1.0	1.0	0.75
No. of strokes per min	276	268	288	280	264	290
Cross-slope (in. per ft.)	0.65	1.08	0.96	0.79	1.13	0.7
Elevation of refuse end of supporting channels above other end (in.)	3.09	3.22	3.31	5.28	4.13	0.94
Average pulp ratio $\frac{\text{Weight of water}}{\text{Weight of coal}}$	1.69	2.30	1.47	1.55	1.74	1.18

TABLE 5.— *Combined Results Table Tests* 11, 12, 17 *and* 19

Including the Following: Slimes, 1.0 Per Cent. with 16.0 Per Cent. Ash, and Classifier Products, 6.8 Per Cent. with 35.0 Per Cent. Ash from Test No. 16 and 5.8 Per Cent. with 57.9 Per Cent. Ash from Test No. 14

Products	Table Feed, Per Cent.	Classifier Feed Per Cent.	Ash,* Per Cent.	Cumul. Wt., Per Cent.	Cumul.* Ash, Per Cent.
Zones 1– 2 table test 19	7.2	1.6	5.8	1.6	5.8
1– 2 11	10.5	2.1	6.0	3.7	5.9
3– 4 19	14.3	3.2	6.4	6.9	6.1
1– 2 12	9.0	2.0	6.4	8.9	6.2
3– 4 11	18.6	3.7	6.6	12.6	6.3
5– 6 19	14.0	3.2	6.9	15.8	6.4
7– 8 19	15.3	3.4	6.9	19.2	6.6
5– 6 11	16.3	3.3	6.9	22.5	6.6
3– 4 12	18.6	4.1	7.0	26.6	6.7
5– 6 12	12.9	2.9	7.6	29.5	6.8
7– 8 11	18.5	3.7	7.7	33.2	6.9
9–10 19	16.0	3.6	7.7	36.8	7.0
1– 2 17	6.5	1.4	7.8	38.2	7.0
7– 8 12	12.2	2.7	8.1	40.9	7.1
11–12 19	11.9	2.7	8.4	43.6	7.1
9–10 11	10.0	2.0	9.1	45.6	7.2
9–10 12	13.4	3.0	9.4	48.6	7.4
3– 4 17	24.7	5.3	9.6	53.9	7.6
13–14 19	7.4	1.7	9.6	55.6	7.6
5– 6 17	18.6	4.0	10.5	59.6	7.8
11–12 11	10.2	2.0	11.0	61.6	7.9
11–12 12	14.7	3.2	11.3	64.8	8.1
7– 8 17	11.6	2.5	12.1	67.3	8.3
15–16 19	8.3	1.9	12.7	69.2	8.4
9–10 17	6.7	1.4	13.4	70.6	8.5
13–14 11	7.3	1.5	14.1	72.1	8.6
13–14 12	9.7	2.1	14.6	74.2	8.8
11–12 17	8.4	1.8	16.0	76.0	8.9
Slimes classifier test 14		1.0	16.0	77.0	9.0
Zones 13–14 table test 17	8.6	1.9	19.0	78.9	9.3
17–18 19	1.6	0.4	20.5	79.3	9.3
15–16 17	2.9	0.6	21.4	79.9	9.4
15–16 11	5.4	1.1	22.1	81.0	9.6
15–16 12	6.1	1.4	22.7	82.4	9.8
17–18 17	3.1	0.7	23.6	83.1	9.9
Classifier refuse (No. 16)†		1.5	29.3	84.6	10.3
Zones 19–20 table test 17	6.3	1.4	29.8	86.0	10.6
17–18 11	2.0	0.4	29.8	86.4	10.7
19–20 19	1.7	0.4	30.0	86.8	10.8
Classifier refuse (No. 16)†		2.1	33.5	88.9	11.3
Zones 17–18 table test 12	1.0	0.2	37.0	89.1	11.4
Classifier refuse (No. 16)†		3.2	38.7	92.3	12.3
(No. 14)†		1.5	51.0	93.8	12.9
Zones 21–22 table test 17	2.6	0.6	51.0	94.4	13.2
19–20 12	1.5	0.3	52.8	94.7	13.3
Classifier refuse (No. 14)†		1.9	54.9	96.6	14.1
Zones 21–22 table test 19	2.3	0.5	58.9	97.1	14.4
19–20 11	0.9	0.2	62.0	97.3	14.4
Classifier refuse (No. 14)†		2.4	64.6	99.7	15.7
Zones 21–22 table test 12	0.9	0.2	74.9	99.9	15.8
21–22 11	0.3	0.1	79.5	100.0	15.8

* Moisture-free basis.

† Compartments No. 6 of classifier tests divided on curve of washed products as follows:

	Weight, Per Cent.	Ash, Per Cent.
Classifier test No. 14	1.5	51.0
	1.9	54.9
	2.4	64.6
Total	5.8	57.9
Classifier test No. 16	1.5	29.3
	2.1	33.5
	3.2	38.7
Total	6.8	35.0

TABLE 5.—*Combined Results Table Tests* 11, 12, 17 *and* 19.—(*Continued*)

	Specific Gravity	Weight, Per Cent.	Ash,* Per Cent.	Cumul. Wt., Per Cent.	Cumul.* Ash, Per Cent.
Float-and-sink of feed	Under 1.38	70.9	7.5	70.9	7.5
	1.38–1.50	14.6	20.8	85.5	9.8
	1.50–1.70	7.2	35.7	92.7	11.8
	Over 1.70	7.3	67.2	100.0	15.8

* Moisture-free basis.

In the case of the results of the classifier-table combination, therefore, the task is to get the data of laboratory tests into a form to show results actually obtainable in good commercial practice. But the steps necessary to this end should not obscure the basic method of measuring washing results used throughout this paper; namely, a direct comparison of the yields secured by washing with those obtained by carefully made float-and-sink tests of the feed.

Comparison of Washing Results with Classified and Unsized Feeds

The first basis on which to compare the washing of unsized and of classified feeds is relative efficiency. If the efficiencies read from Figs. 1 and 6 are tabulated for various ash contents they appear as follows:

Per cent. ash in washed coal	8.0	8.5	9.0	9.5	10.0	10.5
Table test No. 5, unsized feed, Fig. 1						
Tonnage, 4.9 tons per hr.		39.9	91.5	97.7	98.9	99.2
Table test No. 3, unsized feed, Fig. 1						
Tonnage, 7.8 tons per hr.		15.9	31.6	73.0	91.3	98.8
Combined table tests, classified feed, Fig. 6						
Average tonnage, 10.3 tons per hr.	84.0	91.0	94.2	95.8	96.7	97.1

These figures bring out very distinctly the principal advantage of the classifier-table combination, that of producing washed coals of low ash contents; whereas no 8.0 per cent. ash coal was produced in either of the two tests of unsized coal, an efficiency of 84.0 per cent. was made possible by classification. An almost equally distinct improvement is shown at 8.5 per cent. ash, where the preliminary classification made possible a gain in efficiency from 39.9 to 91.0 per cent. Similarly, we see a distinct improvement due to classification at 9.0 per cent. ash. For ashes under 9.0 per cent. with this coal, preliminary classification of the table feed, therefore has a distinct advantage.

Because the results of classified feeds show higher efficiencies at low ashes, we should expect them to show higher efficiencies at the high ashes also, or at least equal efficiencies; for it would be impossible to make any large gain over the efficiencies around 99 per cent. shown in test No. 5. That the data do not indicate equal efficiencies requires

some explanation. Two classifier products, one containing 6.8 per cent. of the feed with 35.0 per cent. ash and the other 5.8 per cent. with 57.9 per cent. ash, were too small to table. Little could have been gained by washing the second of these products; in fact, it was intended to be a refuse. But float-and-sink tests of the 35.0 per cent. ash product showed an appreciable amount of material recoverable by tabling that might have been added to washed coals above 9.0 per cent. ash. As this product, which is that of compartment No. 6 of classifier test No. 16 (Table 7), is fairly well classified, as will appear later in this paper, one may reasonably suppose that if it had been tabled by itself, the recovery would have been at least equal to that actually obtained in table tests Nos. 3 and 5, where it was a part of an unsized feed. But as there was no means of knowing what might have been accomplished, this product was simply added in with the table products in the order of its ash content. This means, therefore, that the efficiency curve of the classifier-table combination shows lower efficiencies for ashes above 9.0 per cent. than would easily have been obtained had this other classifier product been tabled.

But the high efficiencies obtained in treating unsized feeds at ash contents above 10.0 per cent, indicate that classification could have raised the efficiencies only a trifle at most and that from the standpoint of increased washing efficiency, it would be of little advantage. It had, however, another advantage, which will be the second basis for comparison, that of markedly increasing the capacity of the tables.

The average tonnage of the tables treating classified feeds, as is evident from the figures given with the efficiencies just discussed, is 132 per cent. of that of test No. 3 and 210 per cent. of that of test No. 5. As the efficiencies only in the latter of these tests are at all comparable to those with classified feeds, these tests clearly indicate that for equally high efficiencies the tonnage is more than doubled by preliminary classification.

Although the tonnages of classified feeds are high, they are to be regarded as comparative only and not as indicating the maximum tonnages that may be treated on the table with this system. The difficulty of washing any given coal and the maximum size of particles determine the tonnages that may be handled on the table. If, for example, a washed coal of 10 to 11 per cent. ash had been the aim of the experiments given in this paper, a marked increase in tonnage could have been made, doubtless to 13 to 15 tons per hour. Likewise, if the maximum size of feed had been greater, a higher tonnage would have been possible.

In the above figures for capacity, it will be observed that two tonnages are given for unsized feeds, one at 4.9 and the other at 7.8 tons per hour, while only the figure of 10.3 is shown for classified feed. The marked gain in efficiency at low ash contents made on an unsized feed by reducing the tonnage suggests that a similar gain might be shown in the tests

with classified feeds. But the experiments on table capacity conducted at this station do not necessarily indicate such a result. The table is peculiar in that it works best at only one tonnage; its efficiency drops both at too low and at too high a tonnage. Though these tests of classified feeds furnish no direct data as to what would happen at lower tonnages, the performance of the tables during the tests indicated that, with the possible exception of test No. 19 on the very finest product from the classifier, a decrease in the tonnage would not materially have improved the results.

Commercial Applications of Classification

The preceding discussion has been confined to a comparison of the efficiencies and capacities of the tables with classified and with unsized feeds. In a discussion of commercial applications, it is important to consider the efficiencies that may be expected in practice. Even though most experimenters in coal washing agree that better results are obtained as a rule in commercial plants than in laboratory experiments, in this instance it is worth while to point out definitely some few of the factors that will be more favorable.

Perhaps the largest gain over the laboratory results to be had under commercial conditions will come from improved hindered-settling ratios in the classified products, particularly in that from the fifth compartment of the classifier. During these experiments shortage of coal necessitated arbitrarily making the four classified products intended for tabling of equal size in order to insure sufficient coal for a test on each. If a smaller product of perhaps 15 per cent., in place of 27 per cent., of the classifier feed could have been made in the fifth compartment, and the proportions of the first two or three compartments proportionally increased, its ratios could have been distinctly improved while those of the other compartments would not have been affected materially. This possible improvement depends on the fact that the hindered-settling ratios shown in Tables 6 and 7 are each the average of a series of much higher ratios actually existing at any given level in the main sorting column of the classifier. A sorting column might be thought of as made up of a large number of thin horizontal slices, each having the true hindered-settling ratio of coal, bone and shale. Each product of a six-compartment classifier consists of a number of these thin slices. As the specific gravity and size of particles in succeeding slices on the upper end of the sorting column differ only slightly because of the large proportion in the feed of materials of low specific gravity, a product made up of 25 per cent. of the feed from the upper layers would show only a slight variation between the highest and the lowest slices included and the average ratio would be good. But on the lower end of the column, where the differences between successive

slices is great, the same proportion of the classifier feed will represent a much poorer average ratio. Logically, then, the proper method of proportioning classifier products is on a basis of including an equal number of these thin slices in each compartment; if this procedure is followed there will be larger products in the first compartments of the classifier than in the later ones. The result of being unable to make this distribution during these experiments is shown in Tables 6 and 7. If we take the classified products of compartments 3, 4 and 5 from Table 6 as examples, and put the diameter of coal equal to unity, we get the following results:

Ratios in Classified Products from Classifier Test No. 14, Taken from Table 6

Compartment No.	3	4	5
"Coal," under 1.38 sp. gr	1.00	1.00	1.00
"Light bone," 1.38–1.50 sp. gr	0.74	0.77	0.89
"Heavy bone," 1.50–1.70 sp. gr	0.52	0.53	0.74
"Refuse," over 1.70 sp. gr	0.32	0.32	0.47

It will be seen at once that the ratios in No. 5 are much higher than the others. As all the advantages of classification depend on having low ratios, this fact indicates reduced efficiency in the tabling of this product. But our experiments with other coals give every reason to believe that if in these tests a larger supply of coal had permitted a correct proportioning of the classifier products, a distinct improvement could have been made in the results of the classifier-table combination.

Because the original fifth compartment product of test No. 14 proved very difficult to table efficiently, it was run through the classifier a second time (test No. 16) and an additional product was made containing 6.8 per cent. of the original feed with 35.0 per cent. ash. The improved ratios, as shown in Table 7 (compartments Nos. 1 to 5), resulting from dividing the original fifth compartment product into two classified products, made the table separation, test No. 17, enough better to offset the fact that the new sixth compartment product could not be tabled for lack of material.

Besides the better ratios obtainable by re-proportioning the products of the classifier, there is another respect in which the classified products in a commercial plant would be in better condition to table. During these experiments it was necessary to take each of the products of the classifier and store it, in some instances for three or four weeks, until it could be tabled. This delay allowed time for a part of the shale particles contained in the coal to disintegrate. During the table tests this fine shale found its way into the washed coal, whereas in plant operation it would have been separated into the refuse products.

TABLE 6.—*Classification of Raw Coal in the Six-compartmen Hindered-settling Classifier*

Sizes of Particles are Average Diameters Calculated from Screen Tests

Float-and-sink Fraction	Feed, Mm.	Compartments Classifier Test No. 14*				
		1 & 2, Mm.	3, Mm.	4, Mm.	5, Mm.	6, Mm.
"Coal," under 1.38 sp. gr.	2.98	1.16	2.86	3.40	4.62	5.80
"Light bone," 1.38 to 1.50 sp. gr.	3.13	0.73	2.13	2.62	4.10	5.69
"Heavy bone," 1.50 to 1.70 sp. gr.	3.00	0.53	1.48	1.80	3.43	5.11
"Refuse," over 1.70 sp. gr.	2.76	0.33	0.92	1.08	2.16	3.72

* Proportions shown in Table 3.

TABLE 7.—*Re-treatment of Product from Compartment 5 of Classifier Test No. 14 in the Six-compartment Hindered-settling Classifier*

Sizes of Particles are Average Diameters Calculated from Screen Tests

Float-and-sink Fraction	Feed, Mm.	Compartments Classifier Test No. 16				
			1–5, Mm.		6, Mm.	
"Coal," under 1.38 sp. gr.	4.62		4.81		5.91	
"Light bone," 1.38 to 1.50 sp. gr.	4.10		3.92		5.22	
"Heavy bone," 1.50 to 1.70 sp. gr.	3.43		2.98		4.20	
"Refuse," over 1.70 sp. gr.	2.16		1.94		2.74	

In addition to the improved results obtainable in commercial practice by better classification of the table feed, there is also a possible gain from re-treating a middling from each of the tables. As a similar advantage would have accrued to the tests of unsized feeds, this factor did not enter into the previous comparisons of the tabling of unsized and of classified feeds; but in considering the efficiencies possible commercially, it should be taken into account. Screen-sizing and float-and-sink tests of the products of the tables treating classified feeds showed an incomplete separation on account of the presence in the table feeds of particles of coal finer than the bone and shale particles. This poorly classified material resulted from irregularities in the operation of the classifier and from breakage in handling the products. To complete the separation, the logical procedure is to return a small middling to the classifier feed. This middling, which would serve only to pick up any irregularities in the commercial operation of the classifier or the tables, should be distinguished from a final middling product, which might be made as a low-grade fuel.

In addition to the higher efficiencies possible with the classifier-table combination through re-proportioning the classifier products and through

re-treating incompletely separated products, improvement should certainly come from better adjustments of the classifier and the tables where an unlimited amount of coal and ample time for refinements in adjustments are available. Just how much improvement may be expected in commercial washing from these three sources can only be a matter of speculation. However, if the efficiency in making an 8 per cent. ash washed coal is increased proportionally to other tests in which it has been tried, the one factor of re-classifying a middling will increase the efficiency from 84.0 to 92.0 per cent. It seems reasonable to expect, therefore, that in good commercial practice the over-all efficiency will be close to 95.0 per cent. even in making a washed coal of as low as 8.0 per cent. ash.

The Making of a Middling Product

If a washed coal of low ash content is prepared from this coal, the curves in Fig. 6 show that a final middling product for secondary fuel, as well as one for picking up irregularities in the operation, will be necessary. For example, if a washed coal of 8 per cent. ash is produced, the yield of washed products is 63.5 per cent. If no middling is made, the yield of refuse is 36.5 per cent., which, from the refuse curve of washed products, Fig. 6, is seen to contain only 29.4 per cent. ash. The curve, plotted beside this, which shows the ash content in the corresponding float-and-sink reject to be only 31.6 per cent., indicates that improvements in the washing operation cannot do much to raise the low ash content and that they can decrease only the quantity of refuse. Obviously a product containing only 29.4 per cent. ash would contain too much combustible to be wasted as a refuse; hence, a secondary coal should be made. What the ash content of this must be to have it salable is determined by local market conditions. For the sake of illustration, suppose that, in addition to a market for 8 per cent. ash coal, there is also one for 16 per cent. ash middling: The problem is to find the available yield of this product from the yield-ash curve of washed products in Fig. 6. A point is sought on this curve at a higher yield and ash content than that of the washed coal, such that if the washed coal is subtracted, the average ash content of the coal intermediate between the points will be 16 per cent. By a process of trial, a point at 82.1 per cent. yield and 9.8 per cent. ash is selected that will meet these conditions. There are $82.1 \times 9.8 = 805$ units of ash in the washed coal plus middlings. If $8.0 \times 63.5 = 508$ units of ash in the washed coal are subtracted, the difference is 297 or the number of units of ash in $82.1 - 63.5 = 18.6$, or the per cent. of middlings. The ash content of the middlings, therefore, is 297 divided by $18.6 = 16.0$, which is the ash content desired. These figures will serve to illustrate the method of arriving at the yield and ash content of the middlings available under the conditions proposed.

Operating Advantages of the Classifier-table Combination

In a preceding section of this paper the higher efficiencies in washing and the greater capacities of tables have been pointed out as advantages of classification. It is of interest also to note whether this additional step in washing the coal, which inevitably complicates the washing plant, has any purely operating advantages.

There are three that appear of some importance. First, the classifier irons out irregularities in the "rock" content of the table feed; thus it will help solve one of the most baffling of washing problems, that of maintaining a uniform ash content in the washed coal. Second, the classifier automatically corrects in the table feed a size relationship of coal and refuse particles unfavorable to a separation. Often coal is more friable than are its associated impurities, a condition that results in the refuse particles being coarser on the average than the coal particles from which they are to be separated. Such a coal in the natural unsized state is extremely difficult to table efficiently, but becomes very easy to wash after classification. Third, the classifier increases the range of fine sizes that may be washed effectively by the table, by segregating all of the finest refuse into the feed to one table, where the adjustments can be better adapted to its removal from the washed coal. All of these advantages of classification are of distinct importance to good table operation.

Need for Commercial Trials

There has been no opportunity thus far to study the performance of a commercial operating plant and so to determine what problems will develop that cannot be foreseen in semicommercial tests such as those described in this paper. Everything possible has been done in the laboratory to work out details that might give trouble in a plant; but to anticipate every operating difficulty is impossible. This process involving hindered-settling classification of the table feed is ready to be tried on a larger scale, preferably in a pilot plant, by some company that has a difficult washing problem and needs a washed coal of low ash content.

Conclusions

1. On a coal-washing table there are two separations, one according to specific gravity and the other according to size, which, in treating an unsized feed, oppose each other.

2. Classification of the table feed gives a size-specific-gravity relationship of the particles of coal and refuse such that the sizing action of the table aids the separation according to specific gravity.

3. Classification of the table feed renders the separation in producing washed coals of low ash content appreciably more efficient than that with unsized feeds.

4. Classification approximately doubles the capacity of the table while permitting efficiencies equal to those with unsized feeds.

5. The classifier, by making a refuse product ahead of the tables, smooths out irregularities in the table feed, and so aids in securing a uniform ash content in the washed coal.

Acknowledgments

Among the many men who aided in the conduct of these experiments, the writers wish especially to acknowledge the assistance of M. E. Johnson, K. A. Johnson and C. E. Shaffer.

DISCUSSION

T. Fraser, Pittsburgh, Pa. (written discussion).—The important facts brought out by the work reported in this paper are: (1) When operated on a classified feed, tables will make almost complete recovery of the good coal even when separating at low specific gravities, and (2) the greatly increased capacity on classified feed. The authors have already emphasized these conclusions fully.

Tables will make a 97 to 99 per cent. recovery on unsized feed when the separation is made at around 1.8 specific gravity or higher, so as to remove only the slate. This is the result desired at many operations in comparatively clean beds. However, there are many places in the Appalachian and Middle Western coal fields, as well as in the northwest, where a more difficult problem is encountered. In the East, this situation is probably due more often to difficulty in producing a low sulfur product than to difficulty in getting the ash down. However, whether the difficulty be caused by ash or sulfur, the object, so far as washery performance is concerned, is complete recovery of the float coal with a low gravity separation. There is no reason to suppose that the methods advocated in this paper would not be as effective for one purpose as the other.

Noteworthy examples of this kind of coal in the Appalachian field are furnished by some of the central Pennsylvania coking coals. There are in this section many mines producing coal of exceptionally good coking quality, but carrying from 1½ to 3 per cent. sulfur. This sulfur, while finely disseminated in many cases, is yet largely removable by fine crushing and washing at low specific gravity. At a number of washeries in this field, sulfur reductions of 60 per cent. or more have been accomplished. This will be generally recognized as exceptional performance. At some of these plants that I have investigated, the loss of coal in the refuse is high and the rate of feed to individual machines is low, but these conditions have been accepted as an unavoidable sacrifice in order to obtain the desired product. This would be a fruitful field of research for the authors of this paper.

[For additional discussion, see page 283.]

Re-treating Middlings from Coal-washing Tables by Hindered-settling Classification*

By B. M. Bird† and H. F. Yancey,‡ Seattle, Wash.

(New York Meeting, February, 1928)

One of the problems studied by the U. S. Bureau of Mines in cooperation with the University of Washington has been the re-treatment of table middlings. Hydraulic classification has given the best results. Instead of using the classifier to prepare the feed to the tables, as has been the practice for many years in ore dressing, it is used to separate part of a middling product that cannot be re-treated successfully on the table.

Middlings are desirable in a table plant because the separation according to specific gravity is incomplete; that is, in the products of the table there is always an overlapping of free particles of coal and refuse that can be distinguished with heavy liquids such as are used in the float-and-sink test. The causes of this overlapping may be divided into two groups, those outside the table separation itself and those inherent in the table separation. The first of these groups includes, among other causes, the following: (1) poor adjustments of the table, (2) overloading or underloading the table, (3) irregularities in the quantity and quality of the feed, (4) irregularities in the power and water supply, and (5) a vertical dancing motion of the table deck due to weak foundations. These causes of overlapping are largely unnecessary in a properly designed and constructed plant; but whenever they are present they materially affect the character of the middling product, and hence the problem of re-treatment. The other group includes three additional causes: (6) small differences in specific gravity between coal and impurities, (7) a size-specific-gravity relationship of the coal and refuse particles in the table feed such that the sizing action of the table opposes the separation according to the specific gravity, and (8) shapes of particles that prevent a separation.

The sixth, small differences in specific gravity, may be a very important reason for an incomplete separation. The change in specific gravity from coal to shale is always gradual and the designation of one part as coal and another part as refuse is arbitrary. If some specific gravity, such as 1.45, is chosen as the dividing point between washed coal and refuse,

---

* Published by the permission of the Director, U. S. Bureau of Mines.
† Supervising Engineer, U. S. Bureau of Mines.
‡ Associate Chemist, U. S. Bureau of Mines.

the difference between the particles of highest specific gravity to be included in the washed coal and those of lowest specific gravity to be discarded in the refuse, is obviously less than 0.001 unit of specific gravity. The complete separation of materials differing by so small an amount, and in fact differing by a much larger amount, must be very incomplete. Hence, if the proportion of the raw coal near in specific gravity to the point of separation is large, as when attempt is made to produce a low-ash washed coal from a very bony raw coal, the overlapping between washed coal and refuse will be excessive.

The seventh cause, relative size of coal and refuse particles, is also of great importance. In addition to a separation according to specific gravity on the table, there is also one according to size.[1] Separation according to size tends in general to throw coarse particles into the first zones of the table, and smaller particles into later zones, with little regard to specific gravity. In an unsized feed, therefore, separation according to size opposes that according to specific gravity, because it tends to throw coarse particles of bone and shale into the washed coal and fine particles of coal into the refuse. It is this sizing action of the table which constitutes the best argument for classification of the table feed as used in ore dressing and likewise for using hindered-settling classification to clean up a table middling, as will be apparent in the subsequent discussion.

The eighth cause given for overlapping—shape of particle—except in some rare instances, has the opposite effect; the tendency to flatness usually manifested by bone and shale particles aids the separation, and so it will not be considered in discussing the re-treatment of middlings.

Whenever the overlapping due to one or more of these eight causes affects the efficiency of the separation sufficiently to justify the step, it is desirable to cut a product for re-treatment. The purpose of this paper is to point out the true character of table middlings, to show why hydraulic classification is the logical means of re-washing them and to give results of re-treating them using hindered-settling classification in the process.

Character of Table Middlings in Relation to Re-treatment

One characteristic of table middlings, that of a high content of free particles of "coal" and "refuse," may be seen by referring to Table 1, where the percentages of the different specific gravity components in four middlings are shown. It should be noted that these arbitrarily selected specific gravity fractions have been enclosed in quotation marks, "coal,"

[1] R. H. Richards and C. E. Locke: Textbook of Ore Dressing, 2d Ed., 213–215. McGraw-Hill Book Co., New York, 1925.

B. M. Bird.: Sizing Action of Coal-washing Table. Bureau of Mines *Reports of Investigations* Serial 2755 (June, 1926); and many others.

"light bone," etc., throughout this paper to distinguish them from the more general use of the terms *coal* and *refuse*. The term *refuse* without the quotation marks has been used to designate anything beyond the washed-coal product of the table, even though it is recognized that a middling product may be made. In this table, No. 1 is a short "time" sample, taken in the laboratory, of 5 lin. ft. at the corner of the table; therefore it includes mainly materials inherently due to the type of separation effected by the table. No. 2 is a larger cut from the same table test. No. 3 is a sample taken in the laboratory over a period of 4 hr. when certain irregularities in the feed and in the water supply were allowed to occur to simulate poor practise. No. 4 is a sample of the middling from one of the best table plants in the country, taken over a period of 16 hr. As might be expected, each of these products contains free particles of low-ash coal and of high-ash refuse. No. 4, for example, contains 51.5 per cent. of "coal" with 7.2 per cent. ash and 12.1 per cent. of "refuse" with 61.2 per cent. ash. By difference, what might be considered as a "true" middlings, that is, material in which the coal and shale particles are locked grains, consists of only 36.4 per cent. of the product. As all of the two fractions of lowest specific gravity, making 73.7 per cent. of the product with an average ash content of 11.0 per cent., can be included in the washed coal without raising the allowable ash content, 73.7 per cent. of 13.1 per cent., or 9.7 per cent. of the original raw coal is at stake in selecting the proper method of re-treating this middling. Although the proportion of recoverable coal is larger than usual for this plant this example makes it clear that a separation in which no middling product is made for re-treatment would be very wasteful. Also, it shows that selling this product as a low-grade fuel without attempting to recover its content of marketable coal would be uneconomical.

TABLE 1.—*Float-and-sink Tests of Table Middlings*

Product	1		2		3		4	
	Wgt., Per Cent.	Ash, Per Cent.	Wgt., Per Cent.	Ash, Per Cent.	Wgt., Per Cent.	Ash, Per Cent.	Wgt., Per Cent.	Ash, Per Cent.
"Coal," under 1.38 sp. gr....	15.8	9.5	66.6	7.2	60.8	7.2	51.5	7.2
"Light bone," 1.38–1.50 sp. gr.	22.2	23.5	19.4	19.5	21.4	21.0	22.2	19.7
"Heavy bone," 1.50–1.70 sp. gr.	48.3	36.0	12.0	34.0	14.7	34.3	14.2	35.0
"Refuse," over 1.70 sp. gr...	13.7	66.1	2.0	52.1	3.1	52.2	12.1	61.2
Average ash per cent........		33.2		13.7		15.5		20.5
Per cent. of table feed.......	2.0		27.9		23.8		13.1	

Without doubt there are many table plants where the proportions of coal incompletely separated are smaller than shown in these samples.

However, the absence of coal in the refuse is no sign of the absence of overlapping. Such part of the overlapping as is inherent in the separation must occur wherever the table is used. If there is no coal in the refuse, there must be refuse in the coal.

The other characteristic of table middlings, as might be inferred from the previous discussion of the sizing action of the table, is a size-specific-gravity relationship such that there is a progressive increase in size of particles with increasing specific gravity. Table 2 shows the average diameters, as determined by screen-sizing tests, of the specific gravity fractions in the four middlings from Table 1. Beside these average diameters are ratios calculated by taking the diameter of "coal" as unity. The same relationship of size and specific gravity is apparent in all samples, though the highest ratios occur in Nos. 1 and 2. The ratios of particles in No. 1, for example, from lowest to highest specific gravities, are 1.00, 2.00, 3.91, and 2.38, respectively; the ratios for No. 2 are 1.00, 1.33, 1.76, and 1.06. The decrease in the ratios shown in No. 2 is what would be expected from increasing the proportion of middlings from 2.0 per cent. of the feed in No. 1, to 27.9 per cent. in No. 2. If the proportion of middlings was increased still further, the ratios would obviously approach those originally existing in the feed. These two samples, caught during a short interval of time, represent mainly the size relationships inherent in the table separation.

TABLE 2.—*Average Sizes of Components in Table Middlings*

Product	Average Diameters of Particles*							
	1		2		3		4	
	Mm.	Ratio	Mm.	Ratio	Mm.	Ratio	Mm.	Ratio
"Coal," under 1.38 sp. gr....	0.64	1.00	1.25	1.00	1.54	1.00	2.26†	1.00
"Light bone," 1.38–1.50 sp. gr	1.28	2.00	1.66	1.33	1.80	1.17	2.57	1.14
"Heavy bone," 1.50–1.70 sp. gr.	2.50	3.91	2.20	1.76	2.31	1.50	3.23	1.43
"Refuse," over 1.70 sp. gr...	1.52	2.38	1.32	1.06	2.01	1.31	2.96	1.31
Percentage of feed..........	2.0		27.9		23.8		13.1	

* Samples 1, 2, and 3 resulted from tabling coal passing a 407 Ton-Cap screen (approximately equivalent to a 4-mesh standard sieve).

Sample 4, from tabling coal passing a ¾-in. square screen.

† Average of fraction under 1.30 (1.60 mm.) and 1.38 to 1.50 (2.66 mm.).

Nos. 3 and 4 show pronounced effects from causes outside the table, such as irregularities in quantity and quality of feed, and the like. No. 3, taken over a period of 4 hr. in the laboratory, although consisting of 23.8 per cent. of the feed, shows better ratios than No. 4, representing 16 hr. run in a washery, but including only 13.1 per cent. of the feed. Thus,

the attempt in the laboratory to approximate plant conditions showed fewer effects of poor operation than actually existed in an excellent plant when this sample was taken.

In all of the middlings the progressive increase of size with specific gravity is apparent, except in the "refuse." As experiments with other coals have shown "refuse" particles under normal operation to be the coarsest, this exception is attributed to the fact that in this coal the shale particles, which disintegrate fairly readily in water, fell to pieces during the float-and-sink tests subsequent to tabling. When these samples were originally taken, the "refuse" particles, beyond a reasonable doubt, were the coarsest of the middling components.

In general, then, table middlings are characterized by (1) the presence of a large proportion of *free* particles of "coal" and "refuse," (2) a progressive increase in the average diameter of particles with increase in specific gravity, and (3) the presence of varying proportions of coarse particles of "coal" and fine particles of "refuse" caused either by making a very large middling product or by irregularities in the operation, or by both.

TABLE 3.—*Average Sizes of Components of Classified Products*

Product	Average Diameters of Particles							
	Cell 1		Cell 2		Cell 3		Cell 4*	
	Mm.	Ratio	Mm.	Ratio	Mm.	Ratio	Mm.	Ratio
"Coal," under 1.38 sp. gr.....	2.08	1.00	3.85	1.00	5.17	1.00	6.90	1.00
"Light bone," 1.38–1.50 sp. gr.	1.65	0.79	2.80	0.73	3.73	0.72	5.32	0.77
"Heavy bone," 1.50–1.70 sp. gr.	1.23	0.59	2.10	0.55	2.49	0.48	4 46	0.65
"Refuse," over 1.70 sp. gr....	0.89	0.43	1.29	0.34	1.51	0.29	3.74	0.54

* A refuse product.

Whatever the process used to re-wash middlings, it must be adequate to deal with a product having these three characteristics. The table, which is almost the only device used at present, is manifestly suited only to a part of the task, that of recovering the coarse coal and of eliminating the fine refuse. It has so little latitude in adjustment for any given size of feed that it can do but little toward separating fine coal and coarse refuse originally due to the sizing action of the table. One coal company in the West has had an interesting experience in re-tabling middlings. A middling from three tables is continually returned to the feed bin and re-washed on the same tables. Although this step materially increases the recovery of washed coal from the plant, it gradually builds up a circulating load consisting of materials that do not separate. When this material begins to monopolize a large part of the working area of the table deck, it must be run to waste. Float-and-sink tests of this wasted material, however, show that it contains much free coal that, without the

interference of an adverse size-relationship, might be recovered in the washed coal.

Manifestly the table by itself is inadequate to cope with the problem of cleaning its own middlings. Seeing that the point wherein it fails is that of size, one might reason that screens could be used to supplement the separation, that is, that the fine "coal" and the coarse "refuse" might be screened from table middlings and the intermediate sizes re-washed on the table. This method undoubtedly has much to commend it over the present practice of simply re-tabling. But a still more promising method of aiding the table separation of middlings is available, that of hindered-settling classification. It reduces not merely the adverse size-relationship as does the screen; it actually reverses the groupings of particles according to specific gravity and size. Table 3 shows a typical set of products from a four-cell classifier with ratios computed as before. In each cell the groupings of particles is the reverse of that shown previously by the table, as the following examples show:

	Table Middling No. 1, Ratios	Classifier Cell No. 2, Ratios
"Coal," under 1.38	1.00	1.00
"Light bone," 1.38–1.50	2.00	0.73
"Heavy bone," 1.50–1.70	3.91	0.55
"Refuse"	2.38	0.34

Inasmuch as the classifier ratios indicate that particles of "light bone" having diameters over 0.73 of those of "coal" particles can be completely separated from the "coal" in the classifier, we should certainly expect a 100 per cent. separation, at any desired ash content, of this No. 1 middling, in which the "light bone" particles are twice the size of "coal" particles. Undoubtedly this would be possible if these specific fractions represented any natural divisions in the coal. But they do not; each grades gradually into the other. However, an excellent separation should be expected with the above ratios, even with the small average differences in specific gravity existing between "light bone" and "coal."

Unfortunately middlings practically free of coarse particles of coal and fine particles of refuse are rarely found in practice; as a rule they show results of irregularities in the feed, and the like; hence the classifier, to be successful, must provide some means of taking care of these foreign materials. How it accomplishes this will appear in connection with the discussion of the classifier tests.

Classification Tests of Table Middlings

Two classifiers were used during these experiments, a small tube classifier and a semicommercial unit capable of handling 6 tons per hour.

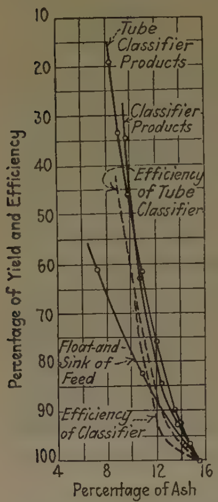


FIG. 1.—CLASSIFIER TESTS OF TABLE MIDDLINGS.

The first of these, referred to later as the "tube," was a simple sorting column 8 in. square with glass windows; it was supplied with a continuous upward current of water. The larger unit, referred to as the "classifier," was the same except that the upper portions of the sorting column were offset so that at each level in the column an upward current of water suited to the character of material at that point could be supplied. The products are numbered in the order of their settling rate from lowest to highest.

The experimental work with the classifiers consisted of tests of the No. 3 middling product shown in Tables 1 and 2, using both the classifier and the tube. The results are shown in Table 4 and in Fig. 1. The efficiencies of the separations made by the classifiers are plotted in the form of curves in Fig. 1. These efficiencies are 100 times the yield of washed coal at any given ash content divided by the yield shown to be present in the feed by float-and-sink.[2]

TABLE 4.—*Classifier Tests of Table Middlings*

Product	Weight, Per Cent.	Ash*, Per Cent.	Cumul. Wt., Per Cent.	Cumul. Ash,* Per Cent.
Tube Test				
Layer 1	18.7	8.4	18.7	8.4
2	14.5	9.8	33.2	9.0
3	12.7	11.8	45.9	9.8
4	15.5	14.1	61.4	10.9
5	14.3	17.1	75.7	12.0
6	13.8	21.5	89.5	13.5
7	10.5	32.2	100.0	15.5
Classifier Test				
Cell 1 & 2	34.1	9.6	34.1	9.6
3	28.2	12.0	62.3	10.7
4	21.8	17.2	84.1	12.4
5	8.5	27.2	92.6	13.7
6	7.4	38.2	100.0	15.5

* Moisture-free basis.

Fig. 1 shows the yields of washed products of the classifier to be higher than those of the tube. As the tube classifier represents the best possible

[2] T. Fraser and H. F. Yancey: Interpretation of Results of Coal-washing Tests. *Trans.* (1923) **69**, 459.

separation by hindered settling, these curves are, of course, misleading. Owing to the fact that the tube test was made somewhat later than the classifier test, it is probable that a larger proportion of the refuse particles, which disintegrate readily, fell to pieces, thus causing the tube to show a lower efficiency. However, these results may be taken to indicate that the classifier will practically duplicate the work of the tube.

At first glance, the efficiencies in Fig. 1 are disappointing, for the classifiers were only 60.0 per cent. efficient at 10.0 per cent. ash. However, these figures include the products of cells 4 and 5, which are composed of particles beyond the hindered-settling ratios of coal, bone, and shale, and due to irregularities in the table operation. These two prod-

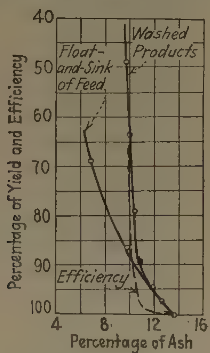


FIG. 2.—EFFICIENCY OF CLASSIFIER AS FINAL CLEANER. CELLS 1, 2, 3 AND 6 = 100 PER CENT.

ucts could be re-washed successfully by the table alone; in commercial practice they would be returned to the table feed or possibly, in a large plant, re-washed on separate tables. In order to determine the efficacy of the classifier as a cleaner alone, these two products, by use of the data in Table 5, were eliminated by calculation from the float-and-sink tests of the feed and from the products of the classifier; the remaining feed and products were each made equal to 100 per cent. These calculations gave the results shown in Fig. 2 and showed the efficiency at 10 per cent. ash to be 77.5 per cent. instead of 60.0 per cent. Although this efficiency in re-washing a middling product may be considered high, it could be bettered in practice, for the efficiencies of both classifiers were undoubtedly reduced by disintegration of the shale particles during the comparatively long interval elapsing in the laboratory between the tabling and classification tests. But if the figures are taken as shown, it will be evident that the classifier has filled an important gap in completing the separation of 69.7 per cent. of the middlings, most of which could not have been re-washed efficiently on the table.

The question now arises as to what efficiencies may be expected in re-washing on tables the products from cells 4 and 5, constituting 30.3

TABLE 5.—*Float-and-sink Tests of Products of Classifier*

Product	Specific Gravity	Weight, Per Cent.	Ash*, Per Cent.	Cumul. Wt., Per Cent.	Cumul Ash, Per* Cent.
Cells 1 and 2	Under 1.38	82.2	6.5	82.2	6.5
Weight per cent. 34.1	1.38–1.50	13.9	20.6	96.1	8.5
	1.50–1.70	3.5	32.9	99.6	9.4
	Over 1.70	0.4	54.0	100.0	9.6
Cell 3	Under 1.38	70.1	7.2	70.1	7.2
Weight per cent. 28.2	1.38–1.50	21.9	19.8	92.0	10.2
	1.50–1.70	7.1	31.2	99.1	11.7
	Over 1.70	0.9	42.5	100.0	12.0
Cell 4	Under 1.38	51.6	8.5	51.6	8.5
Weight per cent. 21.8	1.38–1.50	30.9	21.3	82.5	13.3
	1.50–1.70	15.3	33.6	97.8	16.5
	Over 1.70	2.2	47.6	100.0	17.2
Cell 5	Under 1.38	19.8	10.6	19.8	10.6
Weight per cent. 8.5	1.38–1.50	34.1	22.6	53.9	18.2
	1.50–1.70	38.8	35.2	92.7	25.3
	Over 1.70	7.3	50.7	100.0	27.2
Cell 6	Under 1.38	1.7	12.4	1.7	12.4
Weight per cent. 7.4	1.38–1.50	11.7	24.3	13.4	22.8
	1.50–1.70	66.1	35.9	79.5	33.7
	Over 1.70	20.5	55.8	20.5	38.2
Feed, calculated by combining all cells	Under 1.38	60.8	7.2	60.8	7.2
	1.38–1.50	21.4	21.0	82.2	10.8
	1.50–1.70	14.7	34.3	96.9	14.4
	Over 1.70	3.1	52.2	100.0	15.5

* Moisture-free basis.

per cent. of the middlings. No definite answer is possible because the quantity of these products was too small for tabling tests. However, it is significant that they were fairly well classified, as shown by the following ratios:

	Cell 4		Cell 5	
	Mm.	Ratios	Mm.	Ratios
"Coal"	1.76	1.00	2.70	1.00
"Light bone"	1.42	0.81	2.20	0.81
"Heavy bone"	1.19	0.68	2.09	0.76
"Refuse"	0.90	0.51	1.42	0.53

These ratios, in the reverse order to those shown by the table, would greatly facilitate a sharp separation. Even if part of this advantage should be lost by combining these two products in the feed to one table, a better separation should be possible than in the original unsized condition. However, as the efficiency obtainable is problematical, cells 4 and 5 of the classifier have been regarded as final products in the comparison of the efficiency of the table-classifier combination with that of the table alone.

Fig. 3 shows in graphical form the results of the table test alone, and with re-treatment of middlings. The marked increase in efficiency

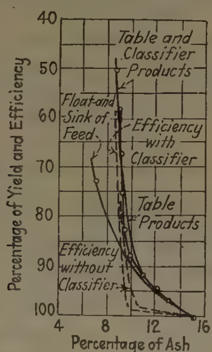


FIG. 3.—COMPARISON OF RESULTS OF TABLING WITH AND WITHOUT CLASSIFICATION OF MIDDLINGS.

between 9.0 per cent. ash and 10.5 per cent. ash due to the re-treatment is shown by the following yields and efficiencies read from Fig. 3:

Ash in washed coal, per cent.	9.0	9.5	10.0	10.5	11.0
Efficiency table and classifier, per cent.	83.7	96.4	98.8	98.9	98.9
Efficiency table alone, per cent.	77.9	88.4	94.5	98.0	98.6
Gain in efficiency, per cent.	5.8	8.0	4.3	0.9	0.3
Gain in yield, per cent.	5.0	7.0	3.8	0.8	0.3

It will be seen from these figures that the gains in efficiencies are largest at the low ashes and that they decrease with increasing ash content; the yields also show this same tendency. Even the 0.8 per cent. gain in yield at 10.5 per cent. ash would be worth the re-treatment in most plants, while the gain of 7.0 per cent. at 9.5 per cent. ash would be profitable in any plant. In practice these figures would show up even better because of the amount of coal recovered by re-tabling cells 4 and 5 from the classifier.

These figures of increased efficiencies and yields should not be considered as out of line with what might be expected in the average table plant; for sample No. 4, from a washery in which the practice was unusually good, was shown to contain roughly five-sixths as much coal as this sample, although representing a cut of only 13.1 per cent. of the feed

as against 27.9 per cent. in the sample re-treated in this test. These increased efficiencies and yields indicate that classification of a table middling might be applied with economic advantage in almost any table plant.

It is to be regretted that no corresponding efficiencies are available for comparison using the table in re-washing this same middling. Sample No. 4 was obtained from a large table plant where the middlings were re-washed on tables with the idea of furnishing this comparison. Unfortunately the head-ash of the sample received for a test did not agree well with the average feed to the re-wash tables in the plant and so, as the data have only a qualitative significance, they have not been included in this paper. However, they indicate that the classifier alone, with no return of any products to the table feed, recovered approximately three times as much middling of the same ash content as the re-wash tables now in use.

Commercial Application

The classifier, particularly the one used in these tests, has certain advantages that favor its adoption in a commercial plant, which may be briefly enumerated as follows: (1) it may be built for any size of plant from one table up; (2) it takes up little space in addition to that required for the tables—about one square foot of floor space per ton per hour capacity; (3) its products are discharged with a loss in head of less than 2 ft., thus permitting the clean products to be run to the same washed-coal elevator as those from tables; (4) it requires little supervision, being practically automatic; (5) it has no moving parts, all of the separation being effected by upward currents of water; (6) the total cost, including that of re-circulating water, should not be above 0.7 cents per ton of middlings washed.

Although tabling with classification of a middling will not give washed coals as low in ash as obtainable by preliminary classification of the feed, nor the increased capacity of the tables under that system, it has certain advantages in simplicity of plant and in its general applicability to existing tabling plants.

Conclusions

1. There is always an incomplete separation of washed coal from other products of the table, which makes desirable some form of re-treatment of a middling product.

2. Table middlings are characterized by the presence of free particles of coal and refuse and by such relationship of size and specific gravity among these particles that the table alone cannot effect the separation.

3. The re-treatment of a middling in the hydraulic classifier increased markedly the yield of washed coal as compared to that of the table alone and made it evident that the classifier might be applied with advantage in almost every table plant.

DISCUSSION

[This discussion refers also to the paper beginning on page 250.]

H. B. Carpenter, Pueblo, Colo. (written discussion).—So far as I know, hindered-settling classification of feed has never been applied to coal-washing tables, although in some washing methods a combination of hindered-settling and washing has been used as a direct means of removing impurities.

Some coals are easily washed, because of their great difference in specific gravity, and the most simple plant will serve, other conditions remaining the same. In a coal containing light bone, separation is very difficult on account of the small difference in specific gravity of the coal substance and light bone.

The authors refer directly to coals of this nature and their adaptation of classification to treating of feed prior to tabling should be of exceptional value to plants that have coals of this kind.

We operate a coal-washing plant in connection with the preparation of coal for our by-product ovens and the character of the coal used compares closely with the coal used in the experiments described by the authors. Our process consists of tabling exclusively and includes rewashing on tables of middling products from the primary tables without primary treatment, and with these problems we have given thought to the application of a classifier as described by the authors, in our washing practice.

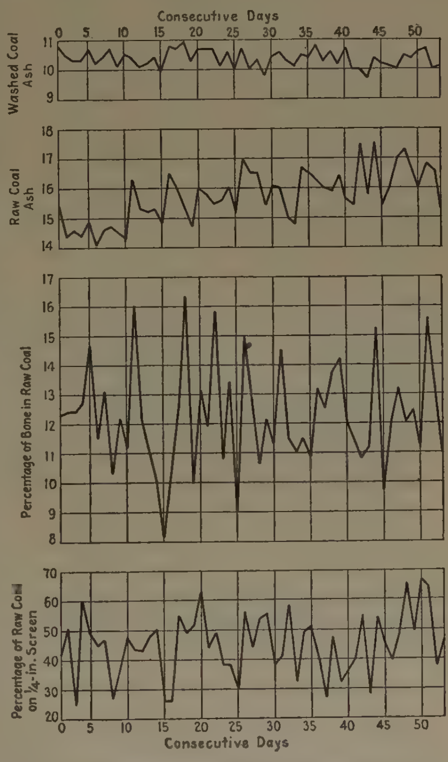


Fig. 4.—Variations in coal over a period of 53 consecutive days.

In our coal-handling equipment we crush the coal to the desired size and, inasmuch as the coal is very friable, every effort is made to prevent the production of an excessive amount of fines. The coal as delivered to our washing tables has a considerably greater range of sizes than the sample used in the work done by the authors, and contains approximately 6 per cent. larger than ½-in. and 40 per cent larger than ¼-in., while the coal tested at Seattle by the authors all passed a No. 407 Ton Cap screen, which gives a product passing through the screen somewhat smaller than ¼-in. mesh. This immediately brings up the question, "What range of sizes can be successfully handled in the classifier?". If this range is small, then our raw coal must be crushed to pass a No. 407 Ton Cap screen with a considerable increase in fines, entailing new problems in sludge handling and drying, or the coal must be screen-sized into two or three sizes before feeding to the classifier, thus making a complex system of distribution to the classifier and to the tables handling the classified product. With a fragile coal, complicated handling and rehandling would doubtless produce an increased amount of sludge with its related problems.

By admitting the possibility of securing a feed of desirable range in size to the classifiers, the problem of compartment products next presents itself. Each compartment product is handled to a table for re-washing and to secure maximum results from the table, the feed must be constant. The classifier is in a delicate stage of equilibrium and a change in either size of feed or the amount of rock and bone present in the feed will cause a change in the amount of product from the various compartments. A storage bin can be provided between the classifier and table to take care of such fluctuations, but such a bin must be of sufficient size to take care of any variation in quantity or the resultant washed coal ash will vary greatly. That a great and sudden variation in the size and quality of the feed to our washing plant does occur, and can hardly be avoided, is evidenced by an examination of Fig. 4, attached. The curves show the variations in the size of coal, the percentage of ash and of bone in the raw coal, and the percentage of ash in the washed product, over a period of 53 consecutive days. A study of these curves indicates a close relation between the size of the coal as fed to the washer and the percentage of bone and ash in the coal. The bone and rock content of our coal is considerably less fragile than the good coal, hence any variation of the amount of impurities in the coal as received will have a decided effect on the size of the crushed coal. It seems impossible, due to the great variation in the quality of coal, to secure an absolutely uniform grade of coal to our washing plant and any apparatus preceding our washing tables which will tend to reduce these variations would result in better washing practice.

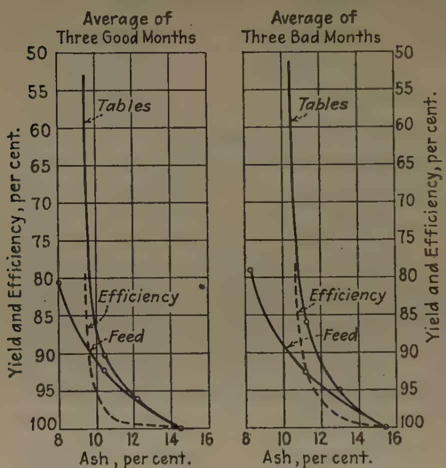


FIG. 5*a*. FIG. 5*b*.

FIG. 5.—CHARACTERISTICS OF COAL FEED AND WASHING PRACTICE OVER A PERIOD OF 3 MONTHS. *a*, QUALILYOF COAL BETTER THAN AVERAGE; *b*, QUALITY WORSE THAN AVERAGE.

There is also a variation from month to month in the quality of the coal washed, which has its resultant effect upon our washing practice. Fig. 5 shows the characteristics of the coal feed and of our washing practice over a period of six months. Fig. 5*a* shows the curves of the feed and the tables for a period of three months, during which we received a better quality of coal than the average, while Fig. 5*b* represents the opposite conditions. The difference in the feed curves is doubtless due to an increase in the percentage of bone and rock present in the feed, which serves merely to move the curve for the bad months from 0.5 to 1 per cent. of ash for the same yields. Now if this were the only change in the coal feed, the table curves should merely move an equal distance with the feed curve, but it will be observed that the table curve for the bad months separates from its companion feed curve more rapidly with a decreasing efficiency. It will also be noticed that the sharp point of separation as evidenced in the efficiency curve of Fig. 5*a* is lacking in the corresponding curve of Fig. 5*b*. This seems to indicate that some other quality of feed has been changed along with the change in the percentage of bone and rock.

In tabling an unsized feed, both size and gravity of the components of the coal play an important part. When a mixture of particles of different size and gravity are shaken together by an action such as that of a washing table, a stratification takes place with the heavier particles tending to go to the bottom and the lighter ones to

the top. Also in such a stratification, the larger particles tend to stay on top, with the smaller going to the bottom. These two factors are in conflict with each other, especially as regards the bony portion of the coal, which, as shown above, is somewhat larger in size than the coal particles. Now, this bone is not heavy enough to cause its weight factor to overcome its size factor, and it therefore rises to the top and is washed off over the riffles with the washed coal. Large rock does not cause this difficulty, as its gravity is high enough to cause it to sink and be carried to the end of the tables by the riffles. This is evidenced by the results given in Table 1 (page 251) and is more pronounced in our own results, due to our greater range of sizes treated. If the classifier will yield compartment products as indicated in the authors' Table 6 (page 268), no larger bone will be present with coal of any given size, and the contamination of the washed coal from this source will be avoided.

The action of the classifier is well illustrated by comparing the table curve in Figs. 2, 3, 4 and 5 (pages 257 to 260) with the curves on Fig. 5 (page 284). It will be noted that the table test on the product from classifier compartment 5, which represents probably the portion of the feed of the highest bone content, and which is shown on Fig. 5 (page 260), shows about 40 per cent. yield under 9.5 per cent. ash, while during our best months 9.5 per cent. ash was the minimum obtained from any table zone. Fig. 6 (page 262) might also be compared with Fig. 5*b* (page 284), as the curves of the feed in both cases are somewhat alike. Our washing practice shows no coal under 10.4 per cent., the curve of Fig. 6 shows 83 per cent. of washed coal produced under 10 per cent. ash, thus illustrating the effect of the removal of bone particles larger than the coal particles from the feed.

One feature of the hindered-settling classifier which is not mentioned by the authors is the ratio of the amount of water to the amount of coal. With an apparatus requiring all products to be floated over a discharge plate, such as this, this ratio will be very high and for the last compartment in which the rock is removed, the ratio may be as high as 50 to 1. The pulp ratio for other compartments will probably average 10 to 1 for a classifier treating material of the size mentioned by the authors, and with a coarser size coal, will probably be greater. The pulp ratio of our washing tables varies between 1 and 1½ to 1 and 2 to 1. From this it will be seen that there is a very large volume of water to be circulated, which, however, will not be very difficult unless this water must likewise be clarified.

Practically all the fines and dust in the coal feed will naturally be concentrated in the product from the first compartment of the classifier, hence probably the water from this compartment alone and the tables handling the product of this compartment will need clarification. This, however, will require a duplex system of water circulation, or one to handle the water which contains the sludge and must be clarified and one to handle practically clear water from other compartments of the classifier. It is difficult to keep two waters separate, especially where any rewashing enters into the process. In all probability, the clean water system would slowly build up with fine coal both from rewashing tables and from drying apparatus and also from the breakage of coal being handled by clean water, and which will require the providing of a clarification system for all the water used in the plant. While the clarification of water to a certain degree suitable for a washing plant is not difficult, some difficult factors of this kind must be considered.

One decided advantage of classification before tabling is in the increased table capacity resulting from primary tabling. According to the authors, the capacity of the washing table is practically doubled when using classified feed. This comparison of course is between a table feed sized through a 407 Ton Cap screen and the same product classified. With a feed product such as ours, or screened through a ¾-in. screen, table capacity should be considerably greater than that secured by Mr. Bird on unclassified feed. The claimed advantage of increased table capacity may

vary with different tables, and hence no absolute comparison can be made. One thought in this connection is that classification before tabling might be an advantage in a high-grade, easily cleaned coal, as this preliminary treatment would reduce the number of tables required to a point where such an installation might prove more economical than straight tabling. Unfortunately, the authors do not make any attempt to state the cost of this classifier, hence any discussion of such an installation from an economical viewpoint is impossible.

While the remodeling of a washing plant from straight tabling to a preliminary classification tabling plant involves some problems, as outlined above, none of these are of sufficient difficulty to make such a change impossible, but the question is, "What benefits would result from such a change?" Naturally, the major benefit is a reduction of ash in the washed coal product. As a basis of comparison we can compare our washing results as shown in Figs. 5*a* and 5*b*, with the authors' results. The float-and-sink curve of this coal compares fairly closely with curves shown in Fig. 6 (page 262), although our coal is probably a little better, as evidenced by the slightly lower ash at all points. No attempt is made to correct these two curves to a common basis, but this difference should react in a slight degree in favor of our coal. A comparison of the table curve of our coal with the tabulation shown in Table 5 (page 263) will give us a comparison table as follows:

Yield, Per Cent.	25	50	60	70	80	85	90	95	100
Ash in washed coal, per cent.:									
Straight tabling	10.4	10.4	10.5	10.6	10.9	11.2	11.8	13.0	15.4
Tabling after classification	5.7	7.5	7.8	8.5	9.4	10.4	11.7	13.0	15.8

A study of this table will indicate that the separation of refuse from the coal is quite clean in the straight tabling process and that it is the prevention of the contamination of the washed coal product by the high-ash bone that is a great advantage of preliminary classification. No washed coal is recovered from any zone of our tables under 10.4 per cent. ash, while in the preliminary classification process one small product is with 5.8 per cent. ash. Thus, if we wish to produce a washed coal of 9.0 per cent. ash, our washing practice would show the following results:

Washed coal.................... 77.0 per cent. at 9.0 per cent. ash.
Boiler coal.................... 15.3 per cent. at 28.8 per cent. ash.
Refuse........................ 7.7 per cent. at 58.1 per cent. ash.

Whether the loss due to this increased amount of boiler coal would be offset by the increased value of our washed coal and coke due to its decreased ash content is a question which I shall not attempt to decide at this time.

I wish to acknowledge the assistance of Mr. Price, Superintendent of Coke Ovens, who compiled some of these data.

Coal Washability Tests as a Guide to the Economic Limit of Coal Washing

By George Stanley Scott,* Wilkes-Barre, Pa.

(New York Meeting, February, 1929)

Many requests for information as to the possibility of washing coals to some predetermined percentage of ash or sulfur have suggested that the producers aim to satisfy some degree of purity set by the user of the fuel, rather than to attain the greatest economic benefit to all concerned.

This paper is written to show that the highest economic purity of washed coal can be determined from washability studies of the raw coal, costs of mining and washing coal, and data on the effect of ash and sulfur on the value of the coal for the particular use for which it is intended.

Coke for Blast-furnace Use

For purposes of illustration, we take the case of coal washed for coke to be used in the iron blast furnace. The method outlined can be made general in its application by substituting for the figures used here the actual operating figures for the particular case under consideration.

In general, it costs the blast-furnace operation from 8 to 25 c. per ton of pig iron to slag out each 1 per cent. of ash introduced into the blast furnace by the coke. This figure depends on which one per cent. of ash is removed, and decreases with decrease in ash. When blast furnaces and coal mines are considered as a whole, the figure decreases to zero and then becomes negative, on account of low recovery of washed coal.

Various figures[1] have been given by different investigators. These figures necessarily differ, not only on account of the differences in operating costs but on account of the difference in the respective fuels as well. For the purpose of this paper, which, as explained, is to point out a method of attacking a certain problem, the figures used in the exposition of the method are of secondary importance. It is apparent that no universally applicable figures can be used, because there are none. Each plant has its own coal,[2] its own operating cost, and its own conditions.

---

* Chief Chemist, American Rheolaveur Corpn.

[1] R. H. Sweetser: Clean Coal. 1927.

T. Fraser: Economy of Well Prepared Coal. *Modern Mining* (1927) **4**, 132.

S. P. Kinney: Blast Furnace Studies at Holt, Ala.

[2] J. R. Campbell: Cleaning Bituminous Coal. See p. 305.

The sulfur introduced with the coke presents a slightly different case but can be reduced to base level and treated in a manner similar to the method proposed for ash. Blast-furnace slag normally will hold, as a maximum, 1.75 to 2.0 per cent. sulfur. Above this, blast-furnace men prefer to take care of the sulfur by enlarging the slag volume. In the case of sulfur, therefore, the cost of removal in the blast furnace is zero until the slag saturation point is reached, when the cost begins to register in real money, and from the saturation point on, the cost for removing each 0.1 per cent. sulfur ranges from 13 to 28 c. per ton of pig iron, depending on conditions.[3]

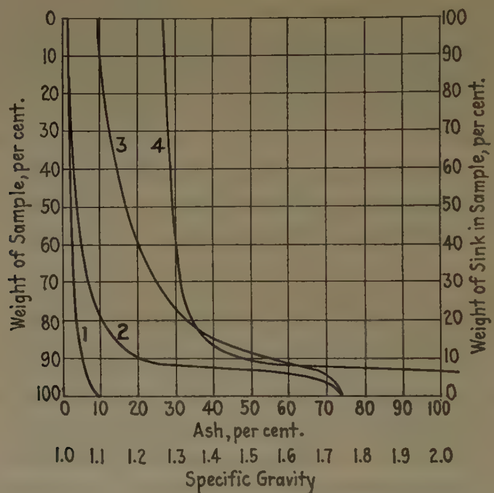


Fig. 1.—Washability curves.

Curve 1. Accumulative coal-ash percentage (float).
Curve 2. Actual ash percentage.
Curve 3. Accumulative slate-ash percentage (sink).
Curve 4. Specific gravity.
Average ash percentage of sample = 9.5.

As in the case of ash, there are no generally applicable figures for sulfur. For different plants there will be points of difference where the slag saturation point is reached, in the actual costs for removal of each 0.1 per cent. sulfur, in mining cost, etc.

It is thus apparent that by removal or reduction of impurities in the coke by preliminary washing of the coal, money can be saved in the blast-furnace operation. Against such savings we must deduct the cost of washing the coal—roughly from 2 to 5c. per ton for each 1 per cent. of refuse rejected in washing (conversion cost plus washing cost). Although the value of a metallurgical fuel increases with purity, the cost of removing impurities by washing also increases, and the highest econo-

[3] T. L. Joseph: Effect of Sulfur on Blast-furnace Process. *Trans.* A.I.M.E. (1925) **71,** 453.

mic purity of washed coal depends on the nature of the coal itself, as shown by its washability curves.

The blast-furnace manager, in order to keep down production costs, naturally demands a fuel as low in ash and sulfur as he thinks he can get at a reasonable cost. The mine manager, on the other hand, wishes to reject as little of his mine output as possible. In the open market, the governing factors will no doubt be supply and demand, but where the coal mines and blast furnaces are owned or controlled by the same corporation, a graphic determination can be made of the percentage of mine output to be rejected in washing, in order that the parent corporation may

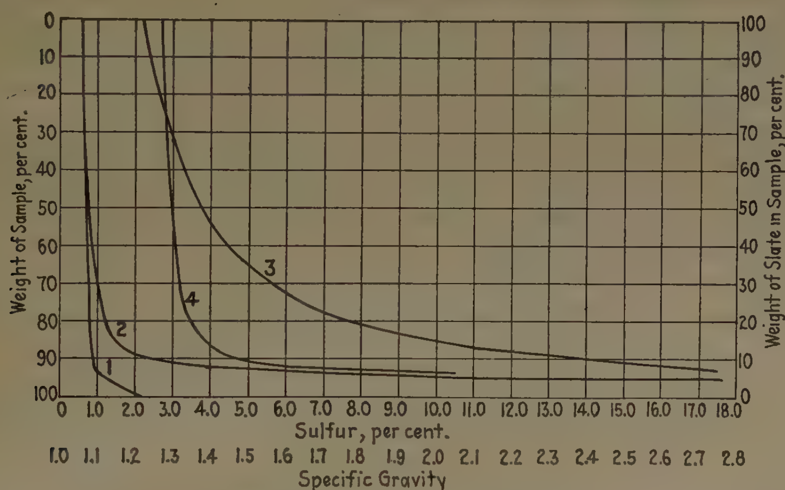


FIG. 2.—WASHABILITY CURVES.

Curve 1. Accumulative coal-sulfur percentage (float).
Curve 2. Actual sulfur percentage curve.
Curve 3. Accumulative slate-sulfur percentage (sink).
Curve 4. Specific gravity.

Average sulfur percentage of sample = 2.20.

realize the greatest financial benefit. The same system could be used by independent companies as a basis for a sliding scale of prices.

Figs. 1 and 2 illustrate the washability curves for the coal selected in illustration of the method. On Fig. 1, curve 1 represents the cumulative ash content of the washed coal, plotted against weight per cent. of the sample, starting from the piece of lowest specific gravity and continuing progressively to the piece of rock or impurity of the highest specific gravity. Curve 2 shows the actual ash of individual pieces according to their varying specific gravity. Curve 3 represents the cumulative ash content of the refuse plotted against weight per cent. of the sample, this time starting with the piece of highest specific gravity and continuing progressively to the piece of lowest specific gravity. Curve 4 shows the

specific gravity of liquid (abscissas) that will float the corresponding weight percentages of the sample (ordinates). Fig. 2 is analogous to Fig. 1, except that sulfur is substituted for ash.[4]

These curves furnish an abundance of information for plant design, plant control, and as to the possibilities of washing. For example, from Fig. 1, 90 per cent. of washed coal could be recovered at an ash content of 4.8 per cent. There would be 10 per cent. of refuse at an ash content of 53.0 per cent. The heaviest material incorporated in the coal and the lightest material incorporated in the refuse would contain 20.0 per cent. ash. The specific gravity of separation would be 1.46. From Fig. 2, the sulfur content of this washed coal would be 0.88 per cent. and the sulfur in the refuse would be 14.0 per cent.

These curves describe a coal that is to be washed, coked, and used as a blast-furnace fuel. What we wish to determine is: What percentage, if any, should be rejected as impurity? The various factors entering the problem may be listed as follows: (1) cost of mining, (2) cost of washing, (3) cost of transportation, (4) cost of coking, (5) value of by-products, (6) cost of taking care of coke ash in blast furnace, (7) cost of taking care of coke sulfur in blast furnace.

TABLE 1.—*Coal and Coke Percentages*

Coal				Coke			
Recovery, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Volatile Matter, Per Cent.	Yield, Per Cent.	Recovery, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.
Raw Coal	9.5	2.20	28.0	74.0	74.00	12.83	1.90
98	8.5	1.75	28.3	73.6	72.15	11.78	1.52
95	6.5	1.23	28.9	73.1	69.45	8.89	1.26
90	4.8	0.88	29.5	72.5	65.27	6.48	0.91
85	4.0	0.82	29.7	72.3	61.48	5.53	0.85
80	3.5	0.80	29.9	72.1	57.66	4.86	0.83
75	3.2	0.78	30.0	72.0	54.00	4.45	0.81
70	3.0	0.74	30.0	72.0	50.40	4.17	0.77
60	2.5	0.72	30.2	71.8	43.09	3.48	0.75
50	2.3	0.70	30.2	71.8	35.90	3.20	0.73
40	2.1	0.68	30.3	71.7	28.30	2.93	0.71
30	1.9	0.66	30.4	71.6	21.50	2.65	0.69
20	1.6	0.64	30.4	71.5	14.32	2.51	0.67
10	1.6	0.60	30.5	71.5	7.35	2.24	0.63

In order to simplify the problem as far as possible, we shall assume that the value of the by-products just pays for the coking operation and therefore eliminates these two factors from our problem. For further simplification, we shall consider costs of mining, cleaning, transportation,

[4] J. R. Campbell: *Op. cit.*, 305–7.

and taking care of the ash in the blast furnace separately in one part and the first factors in connection with the cost of sulfur removal in the blast furnace in a second part.

In Table 1, various analytical figures are set up. The figures for coal are taken directly from the curves, with the exception of volatile matter, which is not shown on the curves. The coke yield shows the recovery of metallurgical coke after elimination of the domestic nut and breeze. The percentage of coal recovered in washing multiplied by the coke yield gives the percentage recovery of coke based on mine output equals 100 per cent. The ash in the coke is calculated by dividing the coal ash by the per cent. yield of coke. The coke sulfur is calculated by multiplying the coal sulfur[5] by 0.64 when raw, and by 0.75 when washed,[6] and dividing the result by the per cent. yield of coke.

We shall now assume a set of blast-furnace data, as follows:

Labor, supplies, service, and overhead, per day	$ 900.00
Limestone, per pound of ash, pounds[7]	1.8
Cost of limestone per ton of 2000 lb	$ 1.50
Furnace capacity, tons of pig iron per day based on ash-free coke	370.00
Rate of coke burning is constant	
Fixed carbon (or 0-ash coke) per ton of pig iron, pounds	1500.00
Pounds of ore per ton of pig iron, raw, natural state	4200.00
Pounds of slag per ton of metal (from ore only)	700.00
Fixed carbon to melt one pound of slag, gasify the corresponding quantity of limestone, and form its proportional share of radiation loss, pounds	0.225
Limestone and ore are both free from sulfur	
There is a $2 freight haul from the mines to the coke ovens, which are adjacent to the blast furnaces.	

Having assumed our coal and blast-furnace data, we can examine some of the economic considerations determined by the ash alone, which are set up in Table 2. It is assumed that the cost of mining a ton of coal and delivering it to the surface is $1.50. It is also assumed that the washing cost is 10 c. per ton of feed coal, and is independent of the amount of reject. For each per cent. recovery of washed coal (column 1) the corresponding cost of the washed product (per ton) is shown in column 2.

---

[5] A. R. Powell: Sulphur in Coal and Coke. *Proc.* Engrs. Soc. Western Pennsylvania (1920) **36**, 622.

[6] J. E. Little in private conversation says: "The ratio of 0.64 for coke sulfur to coal sulfur is rather low for washed coal, from which much of the pyrite has been removed." He suggests 0.75–0.80.

[7] It is assumed that the coal ash consists of aluminum silicate, and that the slag is to contain 50 per cent. lime. The composition of the ash no doubt will change with increasing rejection of mineral matter (See M. C. Stopes and R. V. Wheeler: Spontaneous Combustion of Coal. London, 1927. Colliery Guardian Co.) but it is assumed to be constant in order to avoid undue complexity in our problem. However, in actually applying this method, the composition of the ash should be taken into account at each recovery, as it is very important.

TABLE 2.—*Considerations on Ash*

1	2		3	4[b]	5	6	7	8	9	10
Recovery of Washed Coal, Per Cent.	Cost of Coal per Ton of Washed Coal[a]		Cost of Coke per Ton of Coke	Pounds of Coke	Cost of Coke	Cost of Limestone to Flux Ash in Coke[c]	Extra Overhead Due to Ash in Coke	Sum of Columns 5-6-7	Total Added Cost Due to Ash Only	Value of Each 1 Per Cent. Reduction in Coke Ash
	At the Mines	At the Blast Furnace								
Raw Coal	$1.50	$ 3.50	$ 4.73	1839	$ 4.35	$0.316	$0.550	$ 5.22	$1.67	$0.010
98	1.63	3.63	4.93	1808	4.45	0.285	0.500	5.23	1.53	0.121
95	1.69	3.69	5.05	1720	4.33	0.195	0.360	4.88	1.10	
90	1.78	3.78	5.22	1655	4.32	0.145	0.250	4.71	0.80	0.071
85	1.88	3.88	5.37	1631	4.38	0.119	0.190	4.69	0.66	0.021
80	2.00	4.00	5.55	1613	4.48	0.103	0.180	4.76	0.60	−0.105
75	2.14	4.14	5.75	1603	4.61	0.093	0.170	4.87	0.56	Negative
70	2.28	4.28	5.94	1596	4.74	0.088	0.150	4.98	0.53	Negative
60	2.66	4.66	6.49	1580	5.13	0.073	0.130	5.33	0.46	Negative
50	3.20	5.20	7.24	1573	5.69	0.066	0.120	5.88	0.45	Negative
40	4.00	6.00	8.37	1566	6.56	0.061	0.110	6.73	0.45	Negative
30	5.34	7.34	10.25	1560	7.99	0.055	0.100	8.14	0.45	Negative
20	8.00	10.00	14.00	1557	10.89	0.052	0.090	11.03	0.54	Negative
10	16.00	18.00	25.19	1550	19.50	0.047	0.084	19.63	0.76	Negative

[a] The transportation cost is based on a $2 freight haul.
[b] Data of columns 4–10 are per ton of pig.
[c] It is assumed that limestone can be purchased near the blast furnaces at $1.50 per ton delivered.

This is simply \$1.50 plus \$0.10 divided by per cent. recovery in column 1. The cost of coke per ton (column 3) is the cost of coal per ton divided by the per cent. yield of coke. For example, at 85 per cent. recovery the cost per ton of washed coal is:

$$\text{At the tipple, } \frac{\$1.50 + 0.10}{0.85} = \$1.88$$

$$\text{At the ovens, } \$1.88 + 2.00 = \$3.88$$

and the cost of metallurgical coke per ton is $\frac{\$3.88}{0.723} = \$5.37$. These are the prices to be charged for the various qualities of coke used in the blast furnace.

The quantity of coke required per ton of pig iron produced (column 4) is calculated as follows: First, calculations of furnace charge show that 1500 lb. of fixed carbon are required to produce 1 ton of pig iron from the assumed ore. As the ash in the coke increases from zero, increasing quantities of limestone are required to slag this ash and increasing quantities of fixed carbon (or coke) are required to furnish the necessary heat. The actual quantities of coke required are shown in column 4. The figures are obtained as follows: First, the ore requires 1500 lb. of carbon. This carbon is introduced into the furnace as coke, containing a certain percentage of ash. This ash requires limestone and a certain amount of coke to dissociate the limestone and melt down the slag. This additional coke adds a further quantity of ash, requiring another addition of coke, and so on.

For example, at 85 per cent. recovery of coal, the smelting of the ore requires 1500 lb. of fixed carbon, which is introduced as coke containing 5.53 per cent. ash. The quantity of coke $= \frac{1500}{1 - 0.0553} = 1588$ lb. Of this, 88 lb. is ash, which requires further coke to slag it. This further quantity of coke is equal to:[8]

$$A + AB + AB^2 + AB^3 + \cdots + AB^n = \frac{A}{1 - B} \qquad [2]$$

where

$$A = \frac{2 \times b \times 0.225}{1 - a}$$

and

$$B = \frac{2 \times a \times 0.225}{1 - a}$$

in which

$$a = \text{per cent. ash in coke} \div 100$$

and

$$b = \text{weight of ash in 1588 lb. of coke (88 lb.)}$$

---

[8] Wentworth: College Algebra.

Substituting in equation 2,

$$\frac{A}{1-B} = \frac{0.45b}{1-a} \div \left\{1 - \frac{0.45a}{1-a}\right\}$$

$$= \frac{0.45b}{1-1.45a}$$

$$= \frac{0.45 \times 88}{1 - 1.45 \times 0.0553} = 43 \text{ lb.}$$

The total coke required at 85 per cent. recovery is, therefore,

$$1588 + 43 = 1631 \text{ lb.}$$

The fifth column of Table 2 is simply the cost per ton of coke multiplied by the quantity of coke required. The sixth column is the cost of additional overhead per ton of pig iron, due to decreased capacity of the blast furnace. The sum of columns 5, 6 and 7 cover the fuel cost and the extra flux and overhead costs due to ash in the coke per ton of pig iron (column 8). Inspection of this column shows that the cheapest fuel lies between 85 and 90 per cent. recovery of coal.

Thus the cost of fuel per ton of pig iron, and expense chargeable to fuel at 85 per cent. recovery of coal, is \$4.69. With raw coal the cost would be \$5.22 per ton of pig iron (column 8). The money loss from not washing this coal would be \$5.22 − \$4.69 = \$0.53 per ton of pig iron, considering the ash only.

Column 9 gives the sum of the extra expense for fuel, flux and overhead due to presence of ash in the coke. It is obtained by adding to the sum of columns 6 and 7 the cost of coke above 1500 lb. at the corresponding coke price in column 3.

The figures in column 10 are obtained from Table 1, column 7, and Table 2, column 8, as follows: The difference in coke ash between two adjacent coal recoveries is divided into the difference in blast-furnace fuel costs per ton of pig iron (Table 2, column 8) with the corresponding cokes to give the value per 1 per cent. reduction in coke ash.

Effect of Sulfur

In Table 3, column 1, is the weight of slag produced from the coke ash only. This added to 700 lb. gives the total slag produced (column 2) per ton of pig iron. Column 3 shows the pounds of sulfur introduced into the blast furnace by the coke per ton of pig iron.

Assuming that blast-furnace slag will carry safely 1.67 per cent. of sulfur, in column 4 are given the necessary weights of slag per ton of pig iron to meet this condition.

In the case of raw coal, 1188 lb. (2360–1172) of additional slag must be made to carry off the sulfur. This quantity drops as the sulfur content of the coke decreases, and at 90 per cent. recovery (in the case of

our assumed conditions) no increase in slag volume is required and the blast-furnace cost due to the presence of sulfur vanishes, and is zero from 90 per cent. recovery to 0 per cent. recovery.

TABLE 3.—*Effect of Sulfur*[a]

Recovery, Per Cent.	1 Slag from Coke Ash, Pounds	2 Total Weight of Slag	3 Coke Sulfur, Pounds	4 Weight of Slag to Contain 1.67 Per Cent. Sulfur	5 Fuel Cost Considering Ash and Sulfur	6 Total Added Cost Due to Ash and Sulfur	7 Total Added Cost Due to Sulfur Only	8 Value of Each 0.1 Per Cent. Reduction in Coke Sulfur
Raw Coal	442	1172	39.4	2360	$ 7.65	$4.00	$2.32	$0.192
98	426	1126	27.5	1646	6.82	3.12	1.59	0.381
95	292	1009	21.7	1300	5.48	1.70	0.60	0.220
90	208	914	15.0	898	4.71	0.80	0	
85	175	880	13.9	832	4.69	0.66	0	Negative
80	153	857	13.4	802	4.76	0.60	0	Negative
75	140	843	13.0	779	4.87	0.56	0	Negative
70	131	833	12.3	737	4.98	0.53	0	Negative
60	108	811	11.9	713	5.33	0.46	0	Negative
50	99	801	11.5	689	5.88	0.45	0	Negative
40	91	792	11.1	665	6.73	0.45	0	Negative
30	82	783	10.8	647	8.14	0.45	0	Negative
20	77	778	10.4	623	11.03	0.54	0	Negative
10	69	769	9.8	587	19.63	0.76	0	Negative

[a] Data in Columns 1–8 are per ton of pig.

Column 5 gives the total fuel cost to the blast furnace, considering not only mining, washing, transportation and conversion costs but the effect of the ash and sulfur on the blast-furnace economy as well. With raw coal the cost of fuel is $7.65 per ton of pig iron. At 85 per cent. recovery, the fuel cost is $4.69. Thus, by discarding the heaviest 15 per cent. of material, an economy of $7.65 − $4.69, or $2.96 per ton of pig iron, can be realized.

Graphical Expression of Data

The preceding data may now be summarized in graphic form (Fig. 3).

Curve 1 represents the cost of blast-furnace fuel per ton of pig iron at the various washed-coal recoveries, on the assumption that the coke is ash-free and sulfur-free. The figures are not quite exact, because they were not calculated back to a constant volatile content, which would be the case here, but answer our purpose closely enough.

The presence of ash, however, necessitates additional cost in the blast furnace. This is shown in curve 2.

Curve 3 shows the additional cost to the blast furnace of taking care of the sulfur.

Curve 4 shows the combined effect of all these factors. The cheapest fuel is obtained at 88 per cent. recovery of washed coal. From the shape

of the curve it will be noted that it is safer to err on the side of recovery lower than 88 per cent. rather than on the side of higher recovery.

The final answer to the assumed problem is that 12 per cent. of refuse should be removed from this particular coal by washing. Referring again to Figs. 1 and 2, it will be noted that at 88 per cent. recovery the washed coal would contain 4.40 per cent. ash and 0.86 per cent. sulfur.

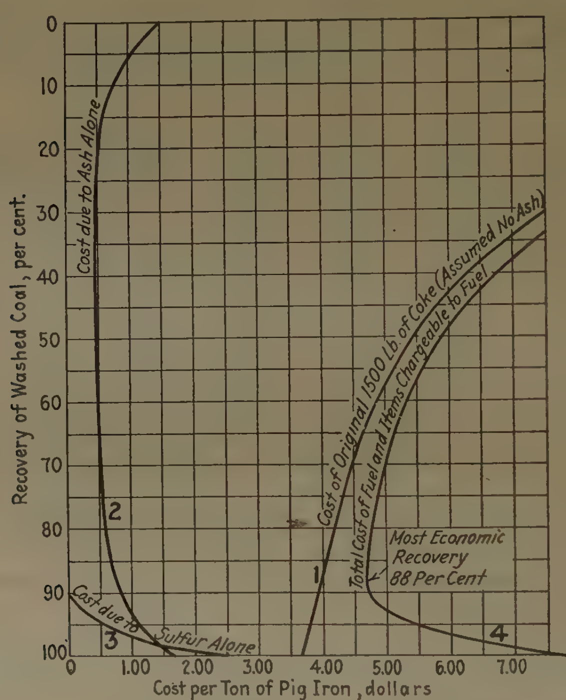


FIG. 3.—SUMMARY OF COST DATA.

Curve 1. Cost of blast-furnace fuel per ton pig iron at various washed-coal recoveries, assuming coke ash-free and sulfur-free.
Curve 2. Additional cost due to presence of ash.
Curve 3. Additional cost due to presence of sulfur.
Curve 4. Combined effect of all factors.

If the figure of 1.3 per cent. sulfur, specified by the A. S. T. M. as a limit, were taken as the mark to shoot at in this case, the recovery of washed coal would be about 95.5 per cent., and the net cost of fuel to the blast furnace $5.55 per ton of pig iron. The cost per ton of coke would be lower at 95.5 per cent. recovery than at 88 per cent. recovery, of course, but the blast furnace would spend the difference and 87 c. more per ton of pig iron in order to keep the sulfur out of the iron. Too good a salesman at the head of either department could, under certain conditions, become an expensive man, with even the best of intentions. Cooperative study

of the coal-washability curves should certainly prove worth many times the cost of the time spent.

Numerous numerical data have been assumed in this paper. It is hoped that those figures approach closely enough to average working conditions to give some idea of the order of magnitude of the various economies involved. It is again pointed out that in this paper the actual numerical values of the figures are of secondary interest only—that the method of applying them is the subject of this paper, and the correct figures for any specific case should be readily obtainable.

Acknowledgments

The author has received many helpful suggestions for this paper from F. F. Marquard and Mr. Shilling of the Carnegie Steel Co., R. H. Sweetser of the American Rolling Mill Co., J. E. Little of the Bethlehem Mines Corpn., J. R. Campbell and John Griffen of the American Rheolaveur Corporation.

DISCUSSION

A. C. Fieldner, Washington, D. C.—This paper should serve as a starting point for an interesting discussion because what we need in this age of tremendously rapid development in coal preparation is a systematic study of methods whereby we can evaluate the different characteristics of a coal for the purpose for which it is to be used. This could be developed further. For example, we should make certain tests on the different float-and-sink fractions. Such tests might include the fusibility of the ash of the different fractions in order to indicate whether the coal after it is washed is going to give more or less trouble from clinkering.

We have been doing some work at the Pittsburgh Experiment Station of the U. S. Bureau of Mines on the fusibility and the composition of the float ash and the sink ash, and the relation of that composition to the unwashed coal. Some coals give a washed product with less fusibility than the raw coal, others with a greater fusibility, depending upon the impurities present.

Professor Parr and Dr. Powell have done some valuable work in showing how to determine the relative amount of organic and pyritic sulfur in coal, important information in coal washing. Recently some work has been done toward devising a method to determine the percentage of fusain, mineral charcoal, in coal, since it may influence the quality of the resulting coal or coke. The paper by Marshall and Bird[9] on agglutinating value shows a further factor that can be used in correlating information on the properties of the coal.

Even the ash of a coal is important. Cobb, in England, has shown that if you add iron oxide to coal, and then make coke from it, its reactivity increases. Other substances have special effects and no doubt the composition of the ash has some influence of this kind. All of these should be tied up with our float-and-sink tests for evaluating coal when studying its washing properties and the utilization of the washed material.

J. B. Morrow, Pittsburgh, Pa.—It is perhaps premature to say too much about the presence of fusain and mineral carbons in coal, although we have done considerable work on it in the last two years. We started this work on the Pittsburgh Seam while making size tests of the coal for washing purposes. These sizes were as follows:

---

[9] See p. 340.

4 by 2, 2 by 1, 1 by ⅜ and ⅜ in. by 0, the latter size being split up into the usual standard screen tests down to 200 mesh. In making these tests we made gravity separations on all the sizes, at three or four specific gravities; that is, somewhere around 1.30 to 1.60 sp. gr. and higher.

In all cases, it was our endeavor to determine the quality of the low-gravity fractions for high-grade metallurgical purposes, the middlings, and the rock end on each size of coal. We not only tested the products for ash and sulfur but also for the fusion point of the ash and the complete analysis of the ash. In other words, we endeavored to find out the true character of the coal seam.

In that investigation we found that in the Youghiogheny River mines the 4 by 2 in. coal at 1.30 sp. gr. had an ash fusion point of approximately 2750° F.; the 2 by 1 and 1 by ⅜ in. coal about the same. On the ⅜ in. by 0 coal at 1.30 sp. gr. we found the ash fusion point was approximately 2250° F., considerably lower than the top sizes. The complete analysis of the ash showed a much higher lime content on the fine coal than on the coarse coal at 1.30 specific gravity.

In order to fix the responsibility for the lower fusion point of the ash in the fine sizes we took actual fusain samples, which at first were a little difficult to find. In testing the fusain, we found that it had a melting point of approximately 2100° F. It was also rather surprising to find that the lime content instead of running about 9 per cent., as it ought to run on 1.30 sp. gr. coal, had increased to 50 or 60 per cent. In other words, the extremely high content of the lime had the natural effect of lowering the ash fusion point. Further investigation showed that when the 28-mesh coal was removed from the ⅜ in. by 0 size, the clean coal had the same fusion point of the ash as the coarse coal. In other words, it was not the character of the coal itself that had anything to do with lowering the melting point.

The presence of fusain, therefore, is an important factor in the by-product oven, making domestic coke, which demands high fusion point ash.

At one operating plant, we are removing the minus 28-mesh coal from the low-gravity coal and putting it into the secondary coal product; that is, we do not put it into the metallurgical coal that is specified to be shipped as a strictly high-grade coal.

The work that has been done in England has also shown that the percentage of fusain taken from the dust of a dry cleaning plant, collected in a Cyclone dust collector, runs as high as 42 to 54 per cent.

One of the chief benefits of a paper like Mr. Scott's is in helping those of us who are endeavoring to clean coal for certain markets and certain uses to find out exactly what the consumer wants, what the value of clean coal is to him; that is, coal free from ash and as low in sulfur as it can be made and with the fusain removed. It is a serious problem we sometimes have to solve. The term "clean coal" is much abused. We have talked and written a great deal about it in the last three years and yet if you ask the very simple question, "What is clean coal?" today, you will probably get as many answers as the people you ask. What does it really mean? Is it coal free from ash or just reasonably free from ash?

For instance, I have seen railroad contracts that said, "The oven producer agrees to furnish to the consumer a coal reasonably free from ash, sulfur and extraneous matter." What is "reasonably free?" Those are the points that we in the producing business would like to know. That is what is wanted and is the reason why a paper like Mr. Scott's is of real value at this time. If we can find out what is actually wanted and how far we can go in coal preparation, there are ways and means of doing it.

There is no question but that we are removing a very large percentage of the fusain, because it is extremely soft and degrades easily into fines below 28 mesh, as the coal goes through the preparation plant. There is another peculiar thing about fusain. We have found that the burning of fusain dust is entirely different from the

burning of a true coal dust. On the construction work of one of our preparation plants we dropped hot rivets on the tipple floor that was covered with 3 or 4 in. of coal dust. The dust became hot and when it was kicked around there was a tar deposit underneath. There was no odor to it, but on testing the stuff we found that there was a concentration of fusain. We have worked up some data showing that in certain parts of the seam the percentage of fusain averages about 3½ per cent., varying from 0.7 per cent. in one place to about 7 per cent. in another.

The subject of moisture in coal is one that everybody seems to like to avoid. We talk about cleaning coal or washing coal but we do not like to talk about moisture. It seems to me that this should be brought out in the open and talked about, just like ash, sulfur or any other constituent. We hear of both the dry cleaning and the wet cleaning plants not being able to operate on account of moisture. The dry cleaning plant naturally has everything in its favor to put out dry coal. If we look into the subject from the standpoint of actual facts, what do we find? One thing we find is that after a spell of wet weather in the Pittsburgh Seam, where the cover is light, the mines are wet. In the spring of the year many of our mines are damp. We have found in the last few weeks that in many places we are actually producing practically as dry a coal from the wet plant as we are from the dry plant, on the size ⅜ in. by 0.

After all, what difference does it make if the coal is cleaned dry or wet if the ultimate moisture in the market product is the same? We have found that on the dry plant the minus ⅜-in. coal varies on samples taken every 2 hr. over a whole month's period, from 3½ up to 8 per cent. moisture, averaging about 5½ per cent. moisture during the spring of the year. On certain bad days, after rain or snow, it will go up to about 6.2 per cent. moisture while at the same time the moisture in the wet plant averages between 5 and 6 per cent. on the same size coal shipped to market. We have actual tests on cars shipped from the wet plant as low as 4.5 to 4.7 per cent. moisture. We have taken the minus ⅜-in. coal from the wet plant in cold weather and have held it in cars for perhaps a week. Then we have taken the cars to the car dump to find out the effect of freezing in the car. We get the wet washed coal down to 5.5 to 5.7 per cent. moisture and after lying in the yard a week, it will dump just as easily as the incoming mine run coal. In cold weather, one of our griefs at the Champion plant, where the coal comes in from five different mines, is the actual dumping of the car, as the hopper will perhaps freeze from 6 to 18 in., sometimes more. If the car has been rained on in transit the water will run down, the top of the car being coned out, and in some cases we have had to hold a mine-run car as long as 12 hr. in order to get the coal out.

One remark was made here about hand preparation as compared with mechanical separation. We have done considerable work on taking samples of clean coal on standard methods of hand preparation. We have found that the hand-prepared clean coal 4 by 2 in. contains from 0.7 up to 5 per cent. of impurities at 1.60 sp. gr. Yet, it all goes on the market as clean coal. Again I ask, "What is clean coal?" Is it coal containing 1, 2 or 3 per cent. of rock or shale? That is something we would like to know and we would like to see standards set up that will be a practical guide to those of us who are in the preparation business and who are endeavoring to serve the consuming public with the highest grade product it is possible to produce under economic conditions.

G. W. Evans, Seattle, Wash.—We have on the Pacific Coast, Vancouver Island, the State of Washington, parts of British Columbia, also in the Crow's Nest Pass District of Alberta, large areas of coal that could not be marketed in the raw state unless they were cleaned.

About four years ago in the Crow's Nest Pass, in a very large deposit of coal, ranging in thickness from 450 to 650 ft. between walls, at Corbin, B. C., the clean

coal and impurities were terribly intermixed. There are five or six large bodies of coal in what is known as Coal Mountain and because the coal was erratic in quality the owners lost the big contract with the Canadian Pacific R. R., the biggest customer in that country. They sent for me to make an investigation. I advised cleaning the coal by water.

We designed a small plant, a trial plant, at a cost of $32,000, and after the plant was built, during a strike, some very able Canadian mining engineers told me that the experiment we were about to conduct was absolutely impractical, that we could not possibly wash coal in the Crow's Nest Pass and get away with it. Temperatures go as low as 55° below zero there and stay there for two weeks at a time. The Canadian Pacific had spent $1,000,000 at Hosmer and a great deal of money at Bankhead. Also, a French company had spent much money at Lisle, in Alberta.

We operated the plant during severe weather of 42° below zero and as a test of shipping the coal (and that is one of the biggest tests), we shipped a carload to Regina, coal that had been washed by water from ¾ in. to 4 in. It was on the track nearly two weeks; the thermometer registered 20° below zero; and when they opened the box of coal at Regina they found that only a little of it had frozen to the doors of the car; the rest was absolutely unaffected.

We had no trouble in cleaning the coal. We had not a particle of trouble in shipping it. We got bold, then, and put in two more units and went down to ¼ in. We washed successfully from ¼ to 4 in. The coal coming from the mine ran between 5 and 6 per cent. moisture, surface moisture, and by by-passing the coal from ¼ in. down, putting through a heat dryer and mixing it back with the washed coal we never had one particle of trouble.

The plant we designed burned down July 3 last year. We have rebuilt the plant but we are going to try out an air table for the small sizes, from about ⅛ to probably ¾ inch.

After observing the air-cleaning plants in four or five places in the Crow's Nest Pass I concluded that air for certain sizes is very inefficient. J. B. Morrow made a trip up there and he and I both saw in one of the dumps of one of the mines 35 or 40 per cent. high-grade combustible coal in the refuse. Of course, that was not the larger sizes. They have corrected some of those mistakes by this time. But we have proved absolutely that coal can be washed in temperatures that run as low as 30° to 45° below zero; and with all due respect to my air-cleaning friends, I would not attempt to use air on anything over ½ to ⅝ in. There are probably sizes under certain conditions where air can be used, maybe from ⅝ to ⅜ or to ¼ in., but from ¼ in. down you are simply giving the coal a ride; there is not any real cleaning. At least, that is the experience I have had in the Northwest.

Water must be used to make a separation from there on down, and I would say that with the Carpenter dryers, which I have seen operated in Colorado, and if necessary, in shipping long distances, if the fines are put in heat dryers of some kind (we have two at Corbin now, two A-14 Ruggles coal dryers that are handling from ¼ in. down) coal can be cleaned with water under any conditions, both as to operating the plant and as to shipment, if you want to put in the equipment.

C. M. Lingle, Nemacolin, Pa.—I would like to know what Mr. Evans washed that coal with, so that it would not freeze at the temperatures mentioned by him.

G. W. Evans.—Water, but properly handled.

G. S. Scott.—Some of you undoubtedly think that this paper is of the grade of an elementary textbook. I agree with you, but that does not mean that it has to stop there.

Practical coal washability curves can be worked out and additional factors included, as the specific case may demand. Two or three people have mentioned the

moisture factor. That is a question which I believe can some day be measured and put right in with a set of cost data such as we have in this paper. Until that time, it should be considered as a qualitative proposition. I believe this method will serve as a starting point for calculating how to get the most money out of a raw coal.

A. ALLEN, Chicago, Ill.—The discussion this afternoon has emphasized the advantage of uniformity in the product of a cleaning plant. The ash and sulfur content in most cases is not so important as its uniformity day after day and month after month. Furnace practice and coking practice must both be based on a uniform standard of quality. The absolute requirements as to ash, sulfur, volatile, etc., can be obtained by mixing provided the quality of each ingredient is uniform.

With this idea in view, we recently worked out a problem for the Youngstown Sheet & Tube Co., in its effort to obtain a coal of uniform quality from a mine in which ash and sulfur varied greatly in different sections. This preparation was considered essential as a preparatory treatment in connection with a coal-cleaning plant.

In working out this problem, the officials of the mining company had determined very accurately the sulfur content in the different sections of the mine and the scheduled operation of the property is such that the trips recur in a cycle of about 2½ hr. It was, therefore, determined that any device to accomplish a mixing of the coal so as to secure uniformity should take at all times a regular proportion of all coal produced during a 2½-hr. cycle.

The mine capacity on an 8-hr. basis is practically 12,000 tons, and a bin was designed to hold a little more than 6000 tons. In this way storage would be provided for a cleaning plant of 6000 tons capacity to operate 16 hr. per day when the mine is running at the 12,000-ton rate.

In order to provide proper bedding and mixing, the bin was constructed with a sloping bottom mounting feeders at frequent intervals throughout its length. The slope of the bottom was such that the amount of coal contained in the rectangle determined by the maximum ordinate and length would be filled in not to exceed a 2-½-hr. run.

In operation, the coal is laid down to a level line entirely covering the slanting bottom, and when this has been accomplished, the coal is spread in uniform layers throughout the length of the bin by means of a horizontal belt conveyor and an automatic traveling tripper running the length of the bin. Ten feeders on the inclined bottom of the bin are then set to deliver each one-tenth of the product required, and the result is that a uniform proportion from every layer of coal laid down in the preceding 2½ hr. is drawn off at a uniform rate, which should give an absolutely uniform mixture. It is then intended to clean the product down to a satisfactory ash and sulfur content. It may be noted also that this bin will give a uniform mixture as to sizing, which is, of course, almost equally important in handling the washing problem.

Another design has been prepared to meet a similar problem arising in almost any coal-cleaning plant, where it is usually necessary to handle coal produced from the mine at a variable rate of hoisting, and with considerable variation both as to sizing and quality. It is essential to run a cleaning plant of any type at as uniform a rate as possible. Starting and stopping the plant disturbs the cleaning operation and produces bad results. It is, therefore, the function of the raw bin back of the coal-cleaning plant to store sufficient coal to iron out the peaks of production and to feed to the cleaning plant a mixture containing as nearly as possible the same proportion of different sizes and the same quality of coal. In a commercial mine this must be accomplished with a minimum amount of breakage.

In order to solve this problem, we have designed a storage bin of rectangular cross-section inclined upward from the discharge point at an angle which will permit the flow of material down the bottom. The rectangular cross-section is modified at

the discharge point by valleys which bring the coal to a feeder. This bin is filled by a drag conveyor which runs up the inclined top of the bin. If the bin is empty, coal from the drag conveyor simply flows for a short distance down the end of the bin into the feeder and if the bin is full the coal pulls over the coal and rolls down the end of the pile. If the bin is normally kept half full, the reserve will be a floating storage with minimum breakage and a considerable mixing effect in that the coal is laid down in inclined layers on the end of the pile and drawn uniformly from the entire area of the bin.

In this case we have designed the conveyor with a flat V-trough and flights; only the middle portion is open to discharge coal and in that way we avoid most of the abrasion that would occur if the entire width of the conveyor were pulled over the coal pile.

Where it is desired to mix radically different coals, as, for instance, in case of a single cleaning plant for two or more mines, several similar bins can very easily be arranged in series so that each of them will discharge through its own feeder a definite proportion of coal on to a common conveyor which feeds the cleaning plant. This conveyor would of course be inclined at the same angle as the bottom of the bin.

I mention these cases here to show that there is a considerable amount of work being done on the question of uniformity and that it is possible to design a storage and mixing bin of large capacity in which the breakage through dropping the coal and grinding under the weight of the pile will be reduced to a minimum.

G. A. Orrok, New York, N. Y.—In my own practice I have been more interested in the cleaning of coal for use in the making of steam, either in powdered form or in work on a stoker, and what we are after there is the same thing that you are after in the coke oven. You want uniformity of product and uniformity of analysis as nearly as possible.

We have had a great deal of trouble with sulfur and those troubles have come not so much from the total amount of sulfur as from the form in which the sulfur is in the coal. I doubt very much if combined sulfur can be washed out or cleaned from coal by any ordinary process of cleaning. Any sulfur that is in the gangue can probably, if the coal is crushed fine enough, be washed out, but the combined sulfur will probably stay in the coal, and it is usually the combined sulfur that gives us the most trouble. Once in a while when we get a combination of iron sulfide in the coal, that is another thing, but that can usually be washed out.

Variations in the coal that comes to us are remarkable. One mine from which a good many tons were shipped to us every month had been very satisfactory. The coal had been reasonable. The analyses had run along with rather good uniformity for a number of years, then suddenly the coal that came in on one cargo fused up, blew off the grates and went up the chimney in bubbles, glass as it were—little circular glass bubbles something like the cenospheres that we talk about in powdered coal.

We looked at our analyses and there was nothing wrong with them. We took analyses of the seam from top to bottom and nothing was wrong. This thing continued for about two weeks and then it ceased entirely. We have yet to find out the cause of that nonuniformity in that particular seam of coal.

I know of another case where there are three shafts on one seam within a comparatively few miles of each other; in fact, the underground workings on two of these shafts are connected by a tunnel. The coal from the three shafts varies in quality—in quality but not in analysis. The coal from two of the shafts can be burned on stokers with comparatively little trouble. The coal from the third shaft clinkers and gives so much trouble that we receive little coal from that mine.

W. L. Remick, New York, N. Y.—I should like to ask the author whether he has ever standardized on the use of the two terms "yield" and "recovery," and just what

his definition is. There seems to be an unfortunate confusion in the minds of some people as to what those two terms mean.

G. S. Scott.—In the first case, we use "recovery per cent." That is in Table 1, page 290, column 1, "Recovery Per Cent." That is recovery of washed coal, or recovery of clean coal in washing, and that column is the base line. Everything refers back to per cent. recovery of clean coal.

Under coke, I have "Yield Per Cent. of Coal." That is in Table 1, column 5. I believe that corresponds to what is considered generally yield of coke; that is, of 100 tons of coal put in the coke oven, these figures come out as coke.

Then the second column is a sort of invention, "Recovery Per Cent. of Coke." This ties the yield of coke with recovery of washed coal.

In other words, any figure in column 1—for example, 70 per cent. recovery of washed coal: when that 70 per cent. recovery of washed coal is coked, 100 tons of the washed coal will give 72 tons of coke, and those 72 tons of coke or 72 per cent. yield of coke times 70 per cent. recovery of washed coal will give per cent. recovery of coke, 50.4. It is that second recovery of coke which may be a little confusing, and it is used in that sense.

W. Tiddy, Swedeland, Pa. (written discussion).—I am sure we will all agree with Mr. Scott that the problem of coal washing should not be considered from the viewpoint of obtaining a low ash and sulfur alone, but from the economic value of the washed coal compared with the raw coal in use.

In dealing with such a question, as applied to the coke-oven and blast-furnace industry, coal washability tests do assist in determining the nature and value of the washed product, although the method of procedure must necessarily be different for each coal under discussion.

Due to this apparent individuality of coals, I feel that Mr. Scott should have included the following additional features in order to obtain a more complete formula for the economical study of this question.

1i. Effect of increase of volatile matter in washed coal on coking properties—partcularly the high-volatile coals (from 30 to 35 per cent. volatile matter).

Two coals washed to the same ash and sulfur content will not necessarily have the same coking properties as the raw coal from which they were produced. In coking practice it may require in one case an increase in low-volatile coal used in mixture, to make a desirable blast-furnace coke, whereas the other washed coal may give an entirely satisfactory product. With such a condition existing in the industry I consider this point must be given consideration in determining the full economical value of washed coal.

2. The question of increased moisture in washed coal.

This higher moisture content of washed coal not only increases the cost of coal delivered, but in winter months increases labor charges for unloading and at all times has a decided effect on amount of coal charged to by-product coke oven, with the consequent change in heat balance and decreased output of coke and by-products. The latter subject was discussed by Dr. E. Dubois,[10] although he dealt with higher moistures than that which the modern washery is marketing today.

In Dr. Dubois' remarks on the influence of the water content of coal and coke on the gas-making process, the first point considered is the influence of the moisture in the coal on the throughput through the ovens. It was found that the addition of 10 per cent. of moisture to the coal had the effect of increasing the heat consumption

---

[10] E. Dubois: Der Einfluss des Wassergehaltes von Kohle und Koks auf Ofenleis tung und Ofengarantien. *Gas und Wasserfach* (1928) **71,** 793.

to a material amount and also of prolonging the carbonization period from 12 hr. to almost 15 hours.

The influence of the moisture content in the coal on the underfiring process was also considered. It was definitely proved that the moisture in the coal has a very important effect on the underfiring process and hence on the efficiency of the ovens, which will not attain specified efficiency unless the moisture in the coal is properly controlled. Heat must also be consumed in driving out the moisture in the coal and this is wasted heat.

The influence of moisture in coke used for underfiring is also considered and detailed figures are worked out in this case as well as in the two previous cases. It was proved that not only has the moisture in coal a marked influence on the throughput of the oven but also that this influence is twice as potent on the gas output of the ovens. Furthermore, when coal with a high moisture content is used for underfiring the ovens, the built-in producers must be much higher than when gas is used for this purpose. It is also pointed out that the tests made were limited to a maximum content of 10 per cent. of water in coal. Many curves and tabulations are given in the original article.

As Mr. Scott's article primarily deals with coal and coke for blast-furnace use, I have limited my few remarks to this feature of the study.

Cleaning Bituminous Coal

By J. R. Campbell,* Scottdale, Pa.

(New York Meeting, February, 1928)

The need for standardizing methods of arriving at definite conclusions regarding the cleanability of a given coal, and for measuring the performance of coal-cleaning equipment, is constantly increasing. Earlier notes by the author on this subject are amplified in this paper, because he believes that better preparation of bituminous coal is a partial solution of the coal operators' troubles.

Washability Studies

The term "washability" of coal was coined when washing was the universal method of separating coal from its impurities by utilizing their difference in specific gravity. This and analogous terms are still used in their broad sense although pneumatic or air separation is now used to clean coal. These terms are so used in this paper.

Too much time or money can hardly be spent on "feed surveys" to get a "picture" of the coal that is to be cleaned. There have been too many "rule of thumb" methods used. Screen analyses and "sink-and-float" tests on the raw coal enable the coal operator and washery man to predict within reasonable limits the results that may be expected from any efficient cleaning apparatus.

The sizes to be examined to get the picture are optional to the investigator. In modern wet washing, the range is from 4 to 0 in. Hand-picking on properly constructed belts is perhaps the standard for cleaning above these sizes. A good picture of the coal may be obtained by the following screen analysis:

3½ to 1¼ in.
1¼ to ½ in.
½ to 5⁄16 in.
5⁄16 in. to 14 mesh
14 to 28 mesh
28 to 48 mesh

and for very complete data on the fines:

48 to 100 mesh
100 to 200 mesh
and through 200 mesh

* Bituminous Representative, American Rheolaveur Corporation.

Sink-and-float tests are made on all the fractions down to 48 mesh at the different gravities and a composite calculated. The minus 48 mesh is calculated into the cleaned coal results in its original state unless oil flotation is being considered as an adjunct.

The range of gravities used in testing is important and may start at 1.30 and end at 2.00. The lower gravity will show the inherent ash and sulfur in the coal and the higher gravity will determine the character of the rock end. A sufficient number of gravity fractions should be made to determine the character of the intermediates, especially the middling or bone product, which is very often the cause of much grief.

The washability of any given coal is directly determined by the washability curves thus set up. Interpretation of the curves is highly important. (This will be discussed later somewhat in detail. Here it will suffice to point out that sharp breaks in the curves tend to show a good washing proposition at the gravity where the break occurs. The break shows a small proportion of tonnage within a wide gravity range.) Another factor in the washability study, always well recognized, is the effect of crushing for liberating the impurities, especially in metallurgical coal.

In the following typical examination of coal, an attempt has been made to cover all the salient points. For the most part, the set-up is self-explanatory.

Washability Test Procedure

The total sample is quartered so that it can be conveniently handled and is then sized over standard screens. Each of these sizes is then tested separately. A sample of sized material is first put into a liquid having the lowest specific gravity of any of the liquids to be used. This separates the sample into two portions, one floating and the other sinking. The portion that sank in the first liquid is then put into the liquid of next higher specific gravity in the range selected and is thereby again separated into two portions. This procedure is repeated until the last portion of the sample is floated in the liquid of highest specific gravity that is to be used.

Each of the portions separated is washed, if inorganic liquids are used, dried and weighed and a sample of each, after thorough grinding and mixing, is analyzed for ash and sulfur.

The results of the sizing test are tabulated to show the screen analysis with the ash and sulfur content of the respective sizes. (Fig. 1.)

The results of the respective float-and-sink tests are tabulated on the form accompanying each washability curve. (Fig. 2.) The columns giving weight per cent. and ash or sulfur per cent. give direct test data. The cumulative weight and ash or sulfur per cent. columns, both float and sink, are obtained by the usual calculation of cumulatives.

Graphic Picture of Total Sample

In order to get a general idea of the character of the sample, the sizing is plotted graphically as represented by the sizing curve, the ordinates being lineal aperture of screens and the abscissas representing cumulative weight per cent. (Fig. 1.) Likewise, the distribution of float-and-sink fractions in the different sizes is plotted with weight per cent. of gravity fractions as ordinates and the mean point of each size fraction as abscissas. In like manner, the ash and sulfur of each of the sizes is plotted as an ordinate and the mean point of each size fraction as abscissa. As will be seen, this curve gives a very complete picture of the raw coal and indicates where the impurities lie and something of their character.

Washability Curves

The float-and-sink data with analyses are then plotted on the corresponding washability curve with ordinates and abscissas as noted on this form. The curves then represent the following information, regarding either ash or sulfur, with an ash curve taken as an example.

Curve 1, which is the cumulative float ash per cent. curve, represents the variation of ash per cent. according to the recovery.

Curve 2 represents the variation in ash per cent. of the material with variation in gravity at which separation is made.

Curve 3 represents the cumulative sink ash per cent. according to the same recovery as Curve 1.

Curve 4 represents the variation of recovery according to specific gravity.

If a certain recovery is chosen, a horizontal line drawn at the point on the ordinates representing this recovery cuts Curve 1 at a point representing the average ash per cent. of the total float coal; Curve 2 at a point representing the ash per cent. of the heaviest piece of material left in the float coal, and likewise the lightest piece of material left in the sink; Curve 3 at a point representing the average ash per cent. of the total sink; Curve 4 at a point representing the specific gravity required to effect the separation shown by the corresponding points on Curves 1, 2 and 3.

In this set-up of washability studies there is an Elkhorn coal complete in all sizes and all gravities with a composite; on the Pittsburgh or No. 8 seam, a composite from $\frac{5}{16}$ in. to 48 mesh, both ash and sulfur; and on the Lower Kittanning, a composite from 2 in. to 48 mesh, both ash and sulfur. All of the tests were made by the American Rheolaveur Corporation.

The Elkhorn curves (Figs. 2 to 6) show the possibilities of washing unsized coal from 4 to 0 in. The composite (Fig. 5), which goes from 4 to $\frac{5}{16}$ in., gives a picture of what may be accomplished in the way of ash

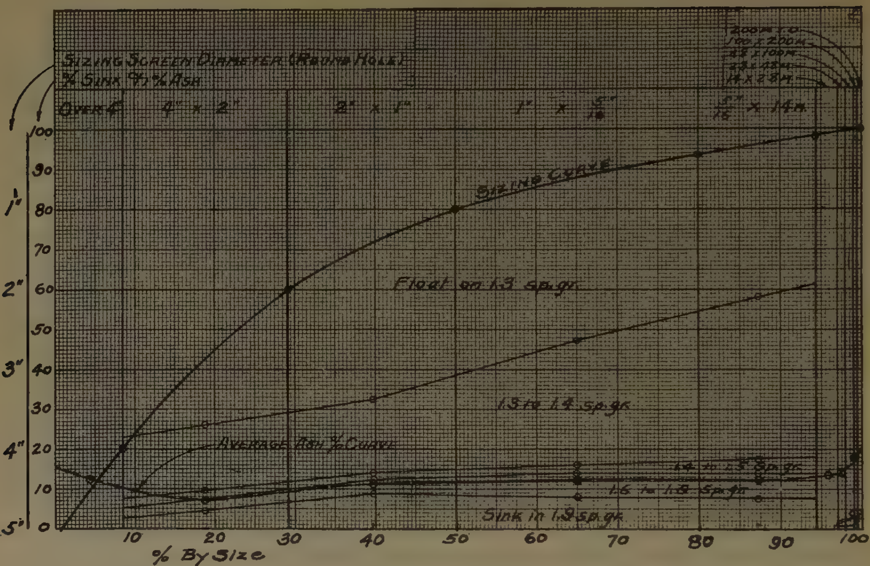


FIG. 1.—WASHABILITY CURVE FOR 4-IN. MODIFIED ELKHORN SEAM, LETCHER COUNTY, KY.

TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

1. Sampling and Peculiarities of Sample:

Coal is fairly hard and tends to break in blocks. The rock is grayish, conchoidal and fairly hard.

2. General Data:

Date sample taken or received:
Total weight of sample: 663 lb.
Moisture per cent.:
Volatile matter, per cent.: 36.5*
Fixed carbon, per cent.:
Sulfur, per cent.:

* On 1.3 to 1.4 sp. gr. fraction of 5/16 in. to 14 mesh size.

3. Sizing Tests:

Denomination	Weight, Lb.	Weight, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Weight Per Cent. of Sample
Over 4 in.	55.75	8.41	12.10		8.41
4 to 2 in.	138.00	20.82	7.24		20.82
2 in. to 0	469.31	70.77	11.64		
Total and average	663.06	100.00	10.76		
2 to 1 in.	36.625	29.42	11.16		20.84
1 to 5/16 in.	52.750	42.36	11.81		29.99
5/16 in. to 0	35.125	28.22	11.85		
Total and average	124.500	100.00	11.64		
Through 5/16 in. Over 14 mesh†	Grams 1399	71.84	10.70		14.32
Through 14 mesh† Over 28 mesh	230	11.82	13.20		2.36
Through 28 mesh Over 48 mesh	137	7.04	13.90		1.41
Through 48 mesh Over 100 mesh	90	4.63			0.92
Through 100 mesh Over 200 mesh	45	2.31	17.40		0.46
Through 200 mesh	46	2.36			0.47
Total and average	1947	100.00	11.85		
Grand total and average			10.76		100.00

† 14 mesh is equivalent to 1–16 in. round hole screen.

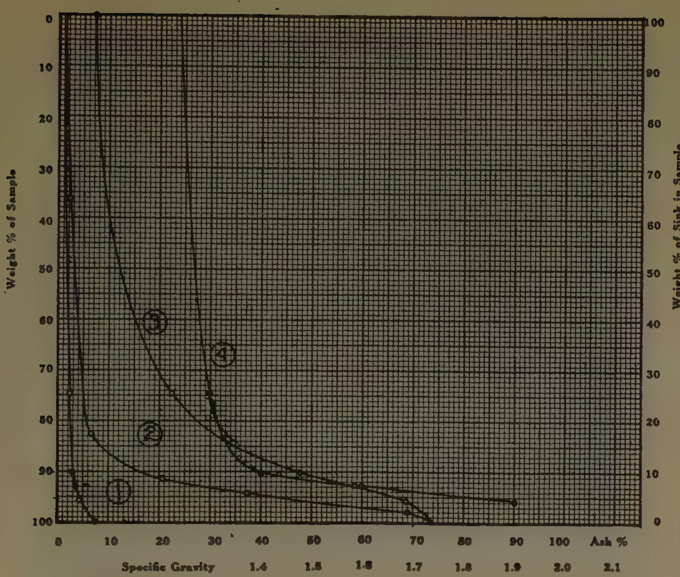


FIG. 2.—FLOAT-AND-SINK TESTS ON 4 TO 2-IN. ELKHORN COAL.
Curve 1, cumulative coal-ash per cent. (float).
Curve 2, actual ash per cent.
Curve 3, cumulative slate-ash per cent. (sink).
Curve 4, specific gravity.
Average ash of the sample, 7.2 per cent.

FLOAT-AND-SINK TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

Ash Results

Size: 4″–2″ Round				Curve No. E-3			
				Cumulative			
				Float		Sink	
Specific Gravity		Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.30		74.9	2.1	74.9	2.1	100.0	7.2
Sink 1.30; Float 1.40........		15.3	6.6	90.2	2.9	25.1	22.6
Sink 1.40; Float 1.60........		2.8	20.5	93.0	3.4	9.8	47.6
Sink 1.60; Float 1.90........		2.3	37.2	95.3	4.2	7.0	58.4
Sink 1.90..................			69.0	100.0	7.2	4.7	69.0
Total......................		100.0					
Average....................			7.2				
Used in Plotting Curve No.	4		2	1, 2* & 4	1	3	3
*The ordinates of Curve No. 2 are the means of each successive pair of float weight percentages with the first ordinate being the mean of first weight percentage and zero.	Abscissas		Abscissas	Ordinates	Abscissas	Ordinates	Abscissas

reduction and recovery. The sharp breaks in the curves show where economical washing should be done. With a recovery of about 90 per cent. the raw coal is reduced from 10.31 per cent. ash to below 4 per cent. ash in the washed coal. The rock end or refuse shows about 62 per cent. ash.

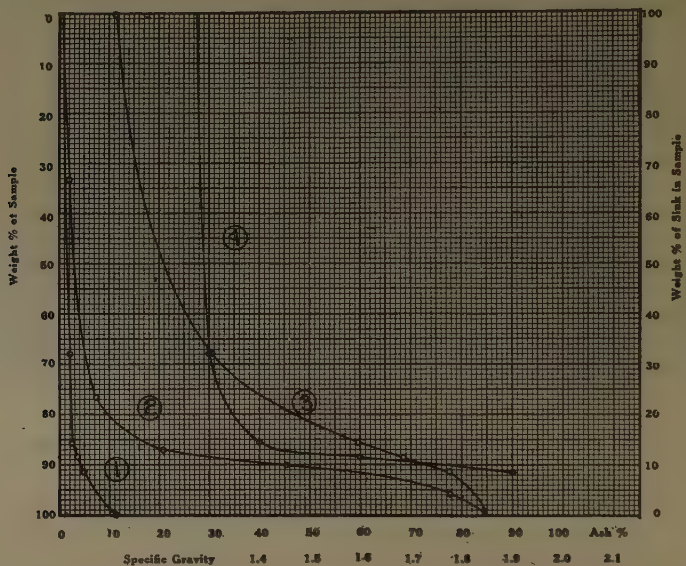


FIG. 3.—FLOAT-AND-SINK TESTS ON 2 TO 1-IN. ELKHORN COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 11.16 PER CENT.

FLOAT-AND-SINK TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

Ash Results

Size: 2″–1″			Curve No. E-4			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.30	67.8	2.0	67.8	2.00	100.0	11.16
Sink 1.30; Float 1.40	18.0	7.2	85.8	2.30	32.2	30.40
Sink 1.40; Float 1.60	2.6	20.9	88.4	3.60	14.2	59.90
Sink 1.60; Float 1.90	3.1	45.2	91.5	5.00	11.6	68.70
Sink 1.90	8.5	77.4	100.0	11.16	8.5	77.40
Total	100.0					
Average		11.16				

Used in plotting as indicated under Fig. 2.

The Pittsburgh, or No. 8, seam of coal (Figs. 7 and 8) is not as amenable to sulfur reduction for metallurgical purposes as some of the lower

productive measures. The example cited shows the effect of fine crushing. The ash curve is satisfactory in this and other curves that have been made on the No. 8 seam. The sulfur curve in this case is not partic-

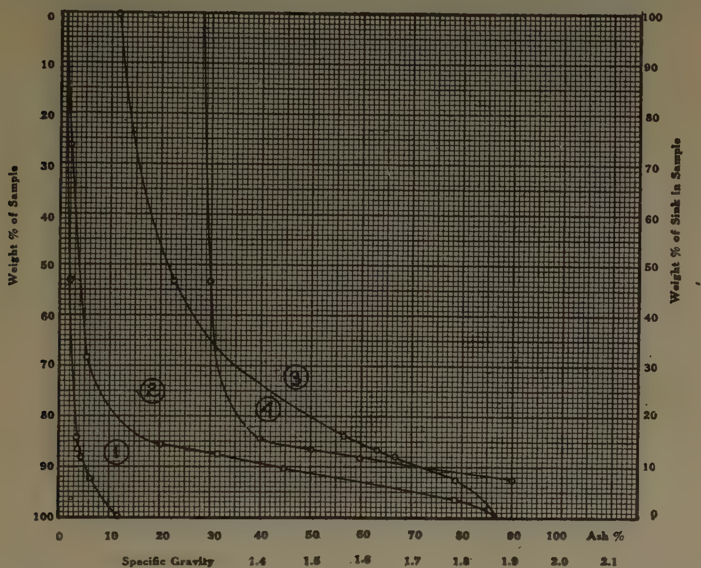


FIG. 4.—FLOAT-AND-SINK TESTS ON 1 TO 5/16-IN. ELKHORN COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 11.81 PER CENT.

FLOAT-AND-SINK TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

Ash Results

Size: 1″–5/16″			Curve No. E-5			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.30	53.0	2.00	53.0	2.00	100.0	11.81
Sink 1.30; Float 1.40	31.0	5.50	84.0	3.30	47.0	22.80
Sink 1.40; Float 1.50	2.4	20.10	86.4	3.80	16.0	56.50
Sink 1.50; Float 1.60	1.5	31.50	87.9	4.20	13.6	63.00
Sink 1.60; Float 1.90	4.2	44.80	92.1	6.10	12.1	66.90
Sink 1.90	7.9	78.70	100.0	11.81	7.9	78.70
Total	100.0					
Average		11.81				

Used in plotting as indicated under Fig. 2.

ularly good. Curve 1 (Fig. 8) is nearly a straight line after the break at about 95 per cent. recovery. It is possible to reduce the sulfur in the Pittsburgh seam from 25 to 30 per cent.; in rare cases, where the initial

sulfur is rather high, reduction has been as high as 35 per cent. For commercial purposes, where ash is the only specification, there is

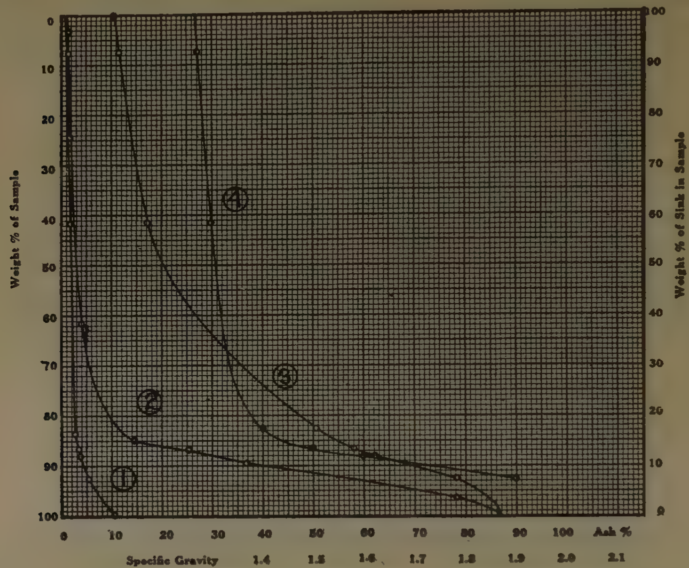


FIG. 5.—FLOAT-AND-SINK TESTS ON 5/16-IN. TO 14-MESH ELKHORN COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 10.7 PER CENT.

FLOAT-AND-SINK TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

Ash Results

Size: 5/16″–14 mesh			Curve No. E-6			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.27.......	7.4	1.50	7.4	1.5	100.0	10.7
Sink 1.27; Float 1.30.......	34.1	1.80	41.5	1.7	92.6	11.5
Sink 1.30; Float 1.40.......	42.1	4.00	83.6	2.9	58.5	17.1
Sink 1.40; Float 1.50.......	2.8	14.60	86.4	3.3	16.4	50.6
Sink 1.50; Float 1.60.......	1.5	25.20	87.9	3.6	13.6	58.0
Sink 1.60; Float 1.90.......	4.7	36.80	92.6	5.3	12.1	62.0
Sink 1.90;	7.4	78.10	100.0	10.7	7.4	78.1
Total....................	100.0					
Average..................		10.68				

Used in plotting as indicated under Fig. 20

no trouble from a washing standpoint. The breaks are nearly always pronounced.

Figs. 9 and 10 show the possibilities of washing the Lower Kittanning seam from both an ash and sulfur standpoint. The curves are character-

istic of both Kittannings, the Upper and Lower. It will be noted that the inherent ash and the inherent sulfur are low. The author always has had great faith in the potential value of the lower measures for metallurgical purposes and has repeatedly mentioned this in talks and writings.

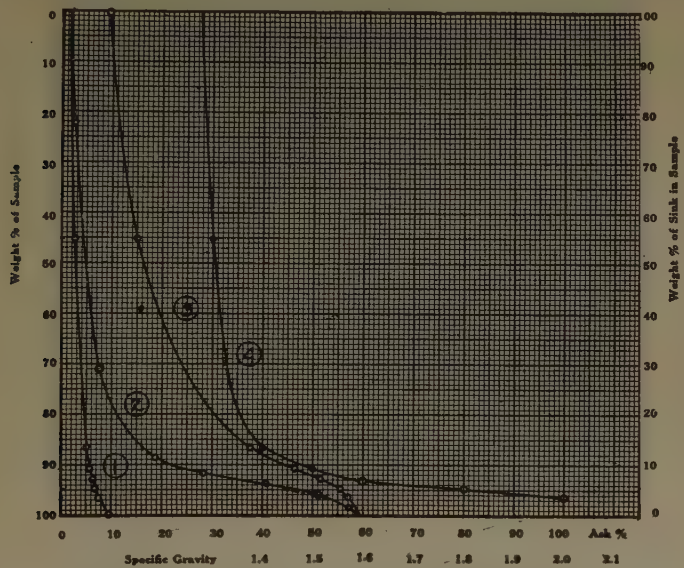


FIG. 6.—FLOAT-AND-SINK TESTS ON COMPOSITE (4 TO 5/16-IN.) ELKHORN COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 10.31 PER CENT.

FLOAT-AND-SINK TESTS ON 4-IN. MODIFIED FROM ELKHORN SEAM, LETCHER COUNTY, KY.

Ash Results

Size: Composite (4″–5/16″)			Curve No. E-2			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.30	63.68	2.04	63.68	2.0	100.00	10.31
Sink 1.30; Float 1.40	22.65	6.10	86.33	3.1	36.32	24.80
Sink 1.40; Float 1.50	1.90	19.27	88.23	3.5	13.67	55.80
Sink 1.50; Float 1.60	1.30	28.00	89.53	3.7	11.77	61.80
Sink 1.60; Float 1.90	3.32	43.40	92.85	5.2	10.47	66.00
Sink 1.90	7.15	76.40	100.00	10.31	7.15	76.40
Total	100.00					
Average		10.31				

Used in plotting as indicated under Fig. 2.

The curves show a composite from 2 in. to 48 mesh, both ash and sulfur. There are breaks in both curves. With a recovery of 90 per

cent., the ash in the washed coal is approximately 6 per cent. and the sulfur 1.03 per cent. with feed sulfur at 2.16 per cent. In this case the sulfur reduction is more than 50 per cent.; on some of the lower productive

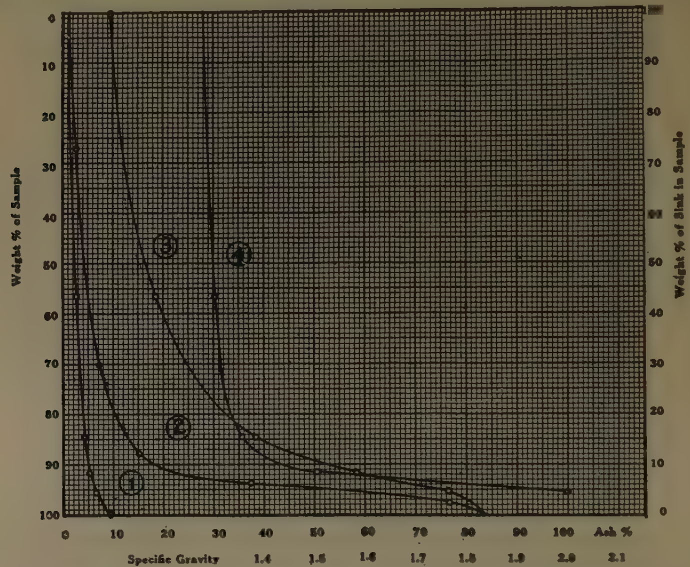


FIG. 7.—FLOAT-AND-SINK TESTS ON 5/16-IN. TO 48-MESH PITTSBURGH COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 9.6 PER CENT.

FLOAT-AND-SINK TESTS ON SLACK FROM PITTSBURGH SEAM, ALLEGHENY COUNTY, PA.
Ash Results

Size: 5/16″–48 Mesh			Curve No. P-3			
Material over 5/16″ Crushed through 5/16″ and Combined with Original Minus 5/16″			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.302	56.51	2.7	56.51	2.7	100.00	9.6
Sink 1.302; Float 1.356	27.62	7.1	84.13	4.1	43.49	18.5
Sink 1.356; Float 1.505	7.49	16.1	91.62	5.1	15.87	38.3
Sink 1.505; Float 2.000	3.92	37.3	95.54	6.4	8.38	58.2
Sink 2.000; Float	4.46	76.4	100.00	9.6	4.46	76.4
Total	100.00					
Average		9.6				

Used in plotting as indicated under Fig. 2.

measures, it runs as high as 60 per cent. The disappointing feature of these coals is the character of the rock end, which is usually rather low in ash.

We have been criticized sometimes for "washing coal on paper with curves" instead of in practical test plants. The author perhaps has made

as many practical tests as any other man in the country. He has found that practical tests do not always bring out all the facts on the washability of a given coal and often lead to wrong conclusions; therefore he

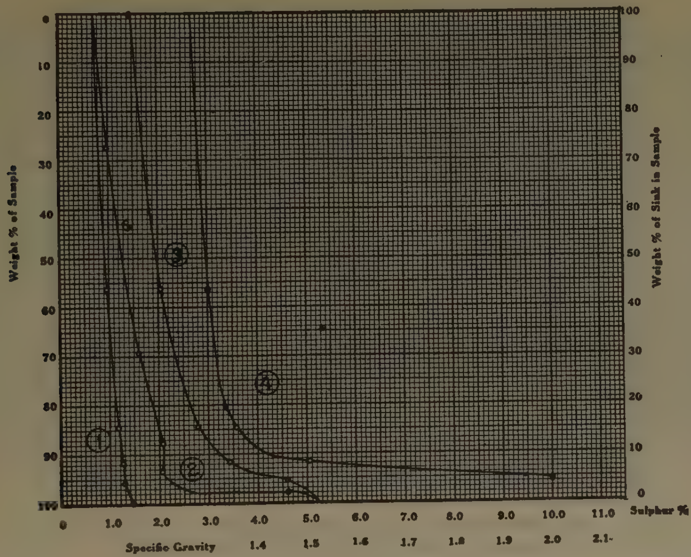


FIG. 8.—FLOAT-AND-SINK TESTS ON 5/16-IN. TO 48-MESH PITTSBURGH COAL. CURVES SAME AS FIG. 2. AVERAGE SULFUR, 1.44 PER CENT.

FLOAT-AND-SINK TESTS ON SLACK FROM PITTSBURGH SEAM, ALLEGHENY COUNTY, PA.
Sulfur Results

Size: 5/16″–48 Mesh	Curve No. P-3S					
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Sulfur, Per Cent.	Weight, Per Cent.	Sulfur, Per Cent.	Weight, Per Cent.	Sulfur, Per Cent.
Float 1.302	56.51	0.98	56.51	0.98	100.00	1.44
Sink 1.302; Float 1.356	27.62	1.60	84.13	1.18	43.49	2.04
Sink 1.356; Float 1.505	7.49	2.09	91.62	1.26	15.87	2.79
Sink 1.505; Float 2.000	3.92	2.06	95.54	1.29	8.38	3.42
Sink 2.000; Float	4.46	4.60	100.00	1.44	4.46	4.60
Total	100.00					
Average		1.44				

Used in plotting as indicated under Fig. 2.

prefers to "wash on paper" first and then see if the washing equipment measures up to the standard. The theory must precede the practice.

Curve 2 is an important factor in determining the washability of any coal. If it flattens out fairly well, the coal will wash easily, but if it comes

down rather straight, the washery will have trouble. This curve indicates the material which, based on the law of probability, will sink or float at definite gravities. It might be called the material in the "twi-

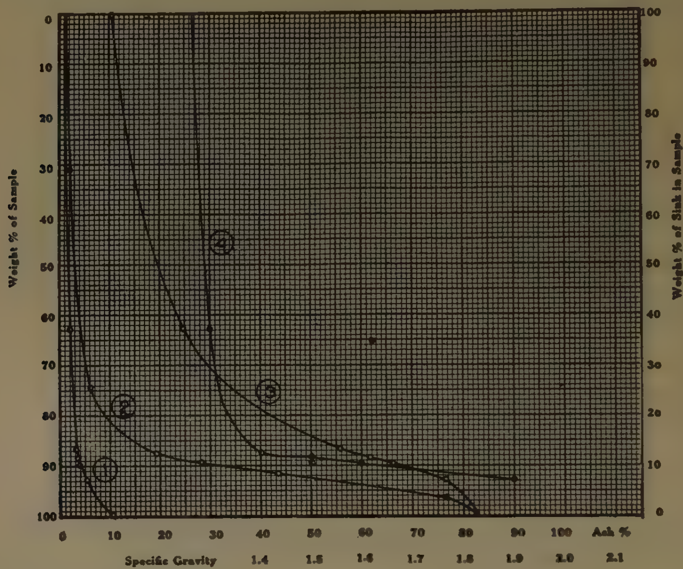


FIG. 9.—FLOAT-AND-SINK TESTS ON COMPOSITE (2-IN. TO 48-MESH) LOWER KITTANNING COAL. CURVES SAME AS FIG. 2. AVERAGE ASH, 9.4 PER CENT.

FLOAT-AND-SINK TESTS ON SLACK FROM LOWER KITTANNING SEAM, CAMBRIA COUNTY, PA.
Ash Results

Size: Composite (2″–48 Mesh)			Curve No. LK-2			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.	Weight, Per Cent.	Ash, Per Cent.
Float 1.30	45.1	2.7	45.1	2.7	100.0	9.4
Sink 1.30; Float 1.40	41.3	7.6	86.4	5.0	54.9	15.0
Sink 1.40; Float 1.50	4.5	18.9	90.9	5.7	13.6	37.3
Sink 1.50; Float 1.60	2.0	28.3	92.9	6.2	9.1	46.4
Sink 1.60; Float 1.80	1.9	40.9	94.8	6.9	7.1	51.5
Sink 1.80; Float 2.00	1.4	51.0	96.2	7.6	5.2	55.4
Sink 2.00;	3.8	57.0	100.0	9.4	3.8	57.0
Total	100.0					
Average		9.4				

Used in plotting as indicated under Fig. 2.

light zone," or the "teeter column." In other words, Curve 2 shows the character of the heaviest particle of material that will remain in the

washed coal or the lightest that will go to the refuse at the various washing gravities or recoveries.

The Pittsburgh, or No. 8, seam of coal, generally speaking, is not as amenable to sulfur reduction as the lower productive measures, wherever

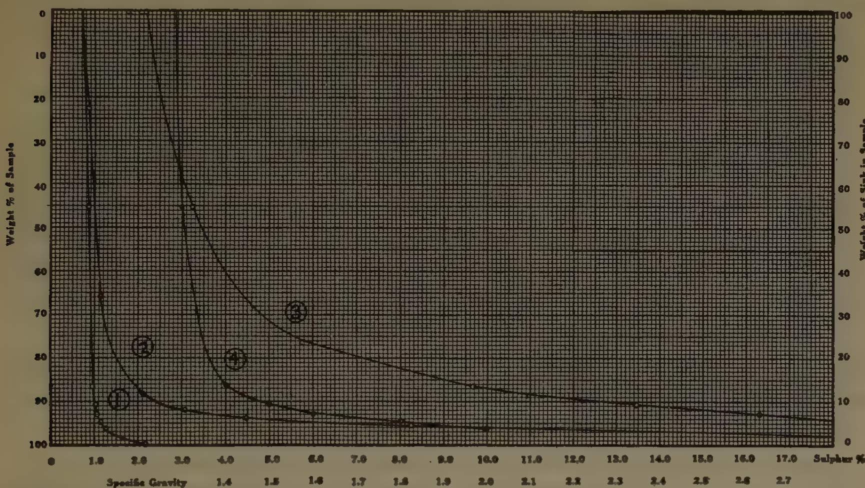


FIG. 10.—FLOAT-AND-SINK TESTS ON COMPOSITE (2-IN. TO 48-MESH) LOWER KITTANNING COAL. CURVES SAME AS FIG. 2. AVERAGE SULFUR, 2.16 PER CENT.

FLOAT-AND-SINK TESTS ON SLACK FROM LOWER KITTANNING SEAM, CAMBRIA COUNTY, PA.
Sulfur Results

Size: Composite (2"–48 Mesh)			Curve No. LK-2S			
			Cumulative			
			Float		Sink	
Specific Gravity	Weight, Per Cent.	Sulfur, Per Cent.	Weight, Per Cent.	Sulfur, Per Cent.	Weight, Per Cent.	Sulfur, Per Cent.
Float 1.30	45.1	0.85	45.1	0.85	100.0	2.16
Sink 1.30; Float 1.40	41.3	1.11	86.4	0.97	54.9	3.23
Sink 1.40; Float 1.50	4.5	2.10	90.9	1.03	13.6	9.67
Sink 1.50; Float 1.60	2.0	3.09	92.9	1.07	9.1	13.44
Sink 1.60; Float 1.80	1.9	4.49	94.8	1.14	7.1	16.30
Sink 1.80; Float 2.00	1.4	8.25	96.2	1.25	5.2	20.70
Sink 2.00;	3.8	25.20	100.0	2.16	3.8	25.20
Total	100.0					
Average		2.16				

Used in plotting as indicated under Fig. 2.

found. The economical gravity for washing this coal seems to be around 1.55 or 1.60; therefore too low a gravity should not be used in testing.

Pocahontas coals usually show good washability curves—very pure coal and very pure rock with a high washing gravity and high recovery. The fines through ⅛ or ¼ in. can usually be by-passed around the washer.

Another class of coal in Central and Western Pennsylvania carries high inherent ash; for instance, 4 to 5 per cent. in the "B" or Miller seam in certain districts. This sometimes occurs in the lower productive measures; the Freeport and Kittannings especially. It usually spells trouble in meeting a low-ash specification with high recovery. The operator should not try to work to too stiff an ash specification where the inherent ash is high.

Sulfur in Coal

The forms of sulfur in coal are generally recognized as organic, pyritic, and sulfate. Simmersbach was an early investigator of these forms. Parr and Powell have made valuable contributions to the literature on the subject. Recently, industrial chemists and engineers, like Russell, of the Youngstown Sheet & Tube Co., and Morrow, of the Pittsburgh Coal Co., have gone into the practical aspects as applied to coal washing.

Morrow[1] has set up some data tending to show that there is, in some cases, a concentration of organic sulfur in the middlings which makes it susceptible of removal in washing. This is contrary to the old idea that organic sulfur is not removed by washing.

Pyritic sulfur is in the form of iron sulfide, pyrite or marcasite (FeS_2). The decomposition, or "weathering" of "sulfur balls" seems to indicate that marcasite is more prevalent than pyrite. This has some bearing in coal washing, in making the circulating, or wash water, acid in character; however, it is a way of sulfur removal by the wet process and may partly account for the use of a factor in determining the practical sulfur results from sink-and-float data.

Formerly, it was believed that FeS_2 was changed to Fe_7S_8, a magnetic sulfide, during the coking process, which eliminated about 43 per cent. of the sulfur. The fact that the "sulfur balls" found in coal become highly magnetic when put through the coking process probably led the early investigators to this conclusion. Now Dr. A. R. Powell[2] writes positively that straight FeS is formed during the coking process, with a deposition of solid sulfur, which makes the new compound magnetic.

Sulfate sulfur is in the form of calcium sulfate and iron sulfate, the latter being caused by oxidation. The percentage is invariably small.

Organic sulfur contains both humus and resinic sulfur; the resinic ranges from 25 to 40 per cent., the humus from 75 to 60 per cent.

The forms of sulfur in the coal apparently have little effect on the percentage "burned out" during by-product coking. The organic and pyritic sulfur seem to behave much alike. The percentage of sulfur volatilized varies from 35 to 45 per cent. from a practical standpoint, depending on conditions. Thus, we may reasonably expect the

[1] J. B. Morrow: Research Displaces Rule-of-Thumb. *Coal Age* (1927) 244.

[2] A. R. Powell: A Study of the Reactions of Coal Sulfur in the Coking Process. *Jnl. Ind. & Engr. Chem.* (1920) **12,** 1069.

sulfur in the coke to be a few points lower than in the raw coal. This ratio is dependent somewhat on the volatile matter in the coal and the consequent coke yield.

Table 1 gives forms and distribution of sulfur in the Pittsburgh, or No. 8, seam of coal in the Pittsburgh District.

TABLE 1.—*Forms and Distribution of Sulfur in the Pittsburgh Seam*

Mine	*A* Per Cent.	*B* Per Cent.	*C* Per Cent.	*D* Per Cent.	*E* Per Cent.	*F* Per Cent.	*G* Per Cent.
Sulfur:							
Organic.....	0.64	0.59	0.56	0.62	0.64	0.71	0.65
Pyritic......	1.26	0.38	1.26	0.36	0.46	1.39	0.74
Sulfate......	0.02	0.02	0.02	0.02	0.02	0.03	0.02
Total.....	1.92	0.99	1.84	1.00	1.12	2.13	1.41

METHODS OF WASHING COAL

The principle underlying the separation of impurities from coal in all important processes is the difference between the specific gravities of coal and refuse. A very practical engineer often referred to the Susquehanna River as the "best coal washer in the world."[3] There are two very active schools for cleaning bituminous coals in this country, (1) pneumatic or dry cleaning and (2) wet washing, with water.[4] Very lately there has sprung up a compromise school, advocating a dual process. The author mentioned the possibilities of a dual system before this Institute as early as 1919, after somewhat careful investigation into the shortcomings of each system as then operated. Figs. 11 and 12 are simple flow sheets of a complete wet-washing plant and a typical dry-cleaning plant.

Read what Forbes[5] has to say: "For water flotation—ability to treat greater variation in size, in other words, to treat as small size as, and larger size coal than, air flotation.

"Ability to treat coal in greater range of sizes in one operation, resulting in less screening required.

"For air flotation—avoidance of added expense on account of added moisture, either for transportation of the extra weight, or for evaporation in use."

---

[3] For a rather complete discussion of the subject, see W. R. Chapman and R. A. Mott: The Cleaning of Coal.-X. *Fuel in Science & Practice* (1927) 6, 15–28.

[4] J. R. Campbell: Coal Washing—Some Factors in the Problem. *Mining Congress Jnl.* (1927) 770.

[5] W. A. Forbes: Technological Problems of the Steel Industry. Amer. Iron and Steel Inst. (1927) 237.

He then gives an example of wet washing, "where, in order to secure as low ash as possible in the coking coal, the bone or intermediate coal was also removed in the washing and used for boiler purposes."

In concluding, he says, "that all of the treating processes will provide a more uniform product by reducing the ash and sulfur content and that a combination of the wet and dry systems would give better results on some coals than either all wet or all dry processes on the same coals. Therefore each installation must be considered separately and the best

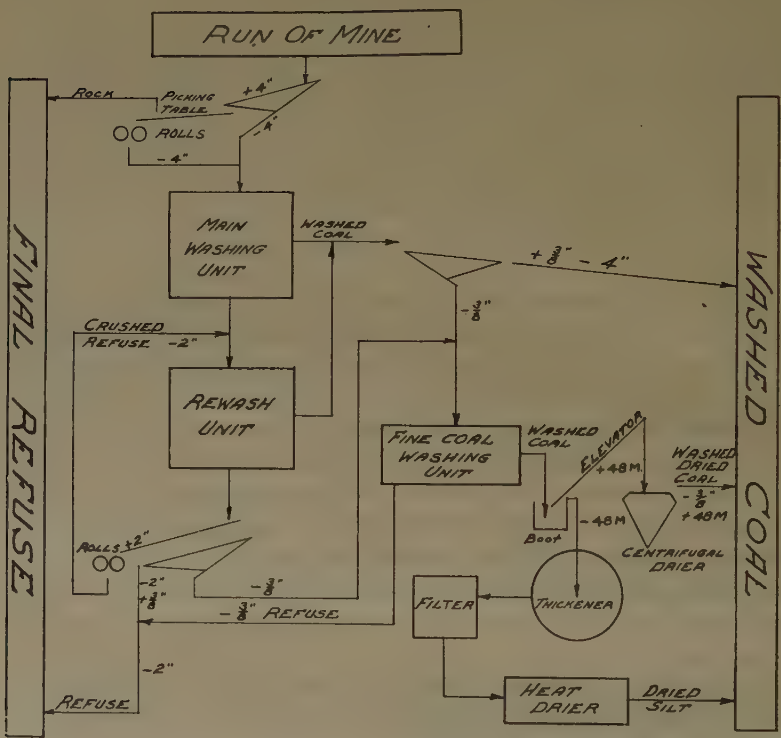
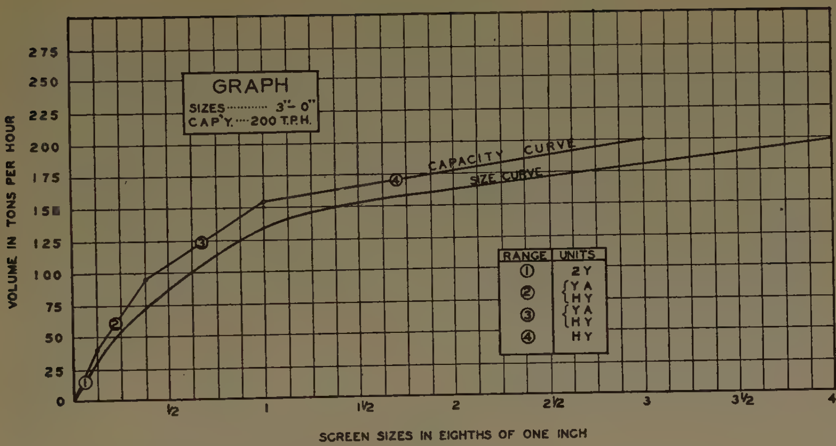


Fig. 11.—Flow sheet of simple wet washery (American Rheolaveur Corpn.)

process selected to obtain the results desired, considering at the same time the added installation and operating costs."

Dry separation is certainly attractive at first sight and probably will always have good application in arid and frigid regions, but its problem is more complex than that of wet washing. The wet system of washing is conducted by various means, such as launder washers; jigs—basket and overflow types; wet concentrating tables; tubs and cones; various types of thickeners and classifiers.



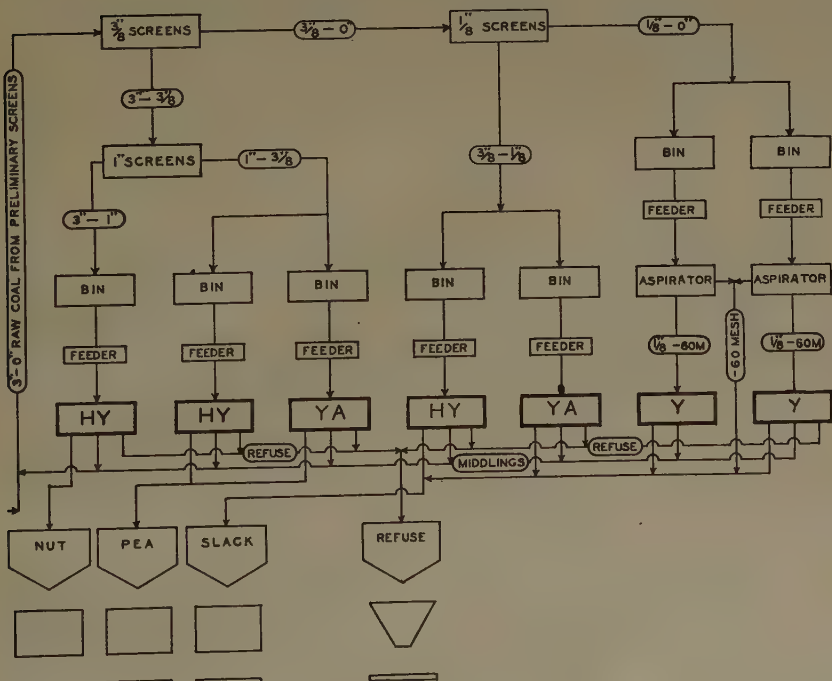


FIG. 12.—FLOW SHEET OF SIMPLE DRY-CLEANING PLANT (AMERICAN COAL CLEANING CORPN.).

Type Y is a full Y solid deck, with fixed transverse slopes; type YA is a split Y deck with adjustable transverse slopes and can treat two different sizes of coal, and type HY is a half-Y, with solid deck and with adjustable transverse slopes but discharges from one side only.

Launder Washers

The "launder washer" naturally heads the list, since it is perhaps the earliest known. The latest development of this type is the Rheolaveur (current washer), which embodies the features of simplified rewashes, refuse drawoffs and a regulating product (or fly wheel), all of which makes it efficient with high capacity. On larger sizes, 3½ to 3⁄64 in., a going plant shows 150 to 200 tons per hr. on a 32-in. launder. On fine sizes, 5⁄16 to 0 in., the rated capacity is 50 to 60 tons per hr. A complete plant under construction in the Pittsburgh district for bituminous coal has a rated capacity of 500 tons per hr. on two 48-in. primary launders.

The launder type of washer, as exemplified by the Rheolaveur, is very successful in Europe, where more than 250 plants are washing about one-third of the coal that is washed. In this country, in a little over two years, the Rheolaveur system of washing has going plants and plants under construction aggregating approximately 10,000,000 tons of anthracite and bituminous coal annually.

Excellent cuts of the Rheolaveur Sealed Level plant for large sizes up to 3½ in. and the free discharge 0 to ⅝ in. are shown in Peele's Handbook.[6]

Jigs

Two types of jigs have carried the burden of bituminous coal washing in this country for a good many years; the basket type and the overflow type. The overflow type may be either single or multiple compartment, and makes its own bed or has an artificial bed of feldspar. The Pittsburgh washing jig is standard for the basket type; the Pittsburgh special and the Foust type are standard for the compartment type.

The Elmore Compartment, or modified type, is used in the Birmingham District, and Forbes[7] undoubtedly refers to results obtained on this type of washer. This jig has good capacity and, for the demands made on it, the results are quite satisfactory.

The most notable installation of the horizontal plunger, overflow type of compartment jig with feldspar on the second compartment, exemplified by the Pittsburgh Special jigs, was the big Middlefork Washery of the U. S. Fuel Co., a subsidiary of the U. S. Steel Corpn. in Franklin County, Southern Illinois. When this was erected, the author was connected with the U. S. Steel Corpn. and knew something of the complex problem presented in washing this high-sulfur coal. From a technical standpoint, based on many sink-and-float surveys of the feed before building the

---

[6] Robert Peele: Mining Engineers Handbook, 2d Ed., 1981–82. New York, 1927. John Wiley & Sons.

[7] W. A. Forbes: *Op. cit.*

plant, the washing results measured up to technical predictions if not to expectations. This type of Pittsburgh jig is adapted to metallurgical and finely crushed coal and has good capacity. The Middlefork plant had 18 jigs and a capacity of 2500 to 3000 tons in 8 hr. on all sizes from 1 to 0 in.

The Foust type jig is illustrated by the installations at the Stag Canyon plant of the Phelps Dodge Corpn. under the direction of J. B. Morrow, an expert practical washery man of this country, and by the Woodlawn plant of the Jones & Laughlin Steel Co. in the Pittsburgh district, washing the Pittsburgh or No. 8 seam of coal. This jig has good capacity—as high as 100 tons per hr. at the Jones & Laughlin plant. This jig is an example of the overflow compartment type, which makes its own bed.

Concentrating Tables

Jigging fine coal presents some difficulties and for that reason considerable attention in late years has been given to wet concentrating tables for handling fines. The principal tables are the Campbell bumping table; the Overstrom table; the Deister-Overstrom table; the Deister Machine "Plato" table. The chief difficulty of the wet tables is their low capacity on fines, varying from 5 to 10 tons per hour.

Bird and Yancey[8] propose to increase both the capacity and efficiency of wet tables by classifying the feed hydraulically instead of mechanically by screens. This is a step in the right direction and it will be interesting to follow the commercial developments and see what this step adds to the cost of tabling.

The wet table is sometimes used as an auxiliary to a jig plant where the fines have to be treated, and the combination makes a good dual plant. Wet tables may also be used as complete installations where fine crushing is necessary.

The Campbell table, as a complete unit, is used by the Bethlehem Steel Co. at its Johnstown plants. This job washes two of the lower productive measures of coal and the maximum size is ¾ in. The Coral plant of the Potter Coal & Coke Co., near Indiana, Pa., has a complete table plant handling 1000 tons per day of Freeport coal. The tables are of the Deister Machine Co. types.

Tubs and Cones

The Robinson-Ramsay tube exemplifies the tub method of cleaning coal, which has come back into favor within recent years for a certain class of work.

The Chance cone uses a fluid mass of sand and water as the flotation medium. So far as known, there is only one installation on bituminous

---

[8] B. M. Bird and H. F. Yancey: Hindered-settling Classification of Feed to Coal-washing Tables. See p. 250.

coal; the Mt. Union plant of the Maderia-Hill & Co. in the Broadtop district, where it handles high tonnage successfully. The fines are by-passed the cone, which is similar to anthracite practice. Paul Sterling tells something about this and other cleaning devices used in the Anthracite Region.[9]

Hydraulic Classifiers

Recently the Menzies Hydro-Separator has made its appearance in bituminous practice, especially in Southern West Virginia. It was originally a development in the anthracite, where it is used on steam sizes. The sponsors for the machine in bituminous practice claim that it will handle all sizes from egg to slack with a capacity of 25 to 40 tons per hr. The egg size is up to 4 and 5 in., which, it is claimed, would eliminate handpicking.

The Hydro-Separator would seem to have considerable application where small units are necessary. It is small and self-contained. The opinion expressed by coal operators who have had tests made on it is that on sized products, with no middlings to speak of, the results are favorable as compared with older processes.

Another classifier is the Hydrotator,[10] which is a development of the Trent tank or thickener. This machine is designed to handle fine coal and we understand there is a bituminous installation at the plant of the Ebensburg Coal Co., Colver, Pa.

The Dorr machines are valuable adjuncts in wet washing.

Dry Cleaning

The accepted methods of dry cleaning are hand picking, mechanical pickers and spirals, Bradford breakers and dry tables. We will concern ourselves only with dry tables.

The author's first experience with dry tables was in Dubuque, Iowa, about 1914, where W. W. Bonson had the first dry coal tables. Later, the author spent considerable time at the parent plant of Sutton, Steele & Steele, in Dallas, Texas, becoming acquainted with the Steele Brothers, Walter L. and Edwin G., and Henry Sutton, the originator of the 3-S table, or S-J as designated on the models. The American Coal Cleaning Corpn., Welch, W. Va., later acquired the rights to use the Sutton patents and built the third dry plant in the country at McComas, W. Va., for the American Coal Co., under the direction of a large construction company. The S-J types of table was used with anti-gravity screens, which were later supplanted by Hummer screens.

---

[9] Paul Sterling: Modern Anthracite. *Proc.* Eng. Soc. N. E. Penna. (Oct. 29, 1927).

[10] Illustrated in paper by W. L. Remick: Fine-coal Cleaning by the Hydrotator Process. *Trans.* (1927) **75**, 570.

The Y table has been developed by the parent company, which has, within seven or eight years, installed dry table plants in this country, Canada, England and Australia, aggregating over 9,000,000 tons annually. Close pre-sizing is needed to obtain the best results. The rated capacity of the Y table is as follows:

Inches	Tons per Hour
3 by 1	70
1 by 3/8	50
3/8 by 1/8	35
1/8 by 0	20

In a previous paper,[11] it has been noted that dry tables have been devised and plants built by Roberts & Schaeffer Co., Chicago, Ill., Heyl & Patterson, Inc., Pittsburgh, Pa.; Peale, Peacock & Kerr, St. Benedict, Pa.

Dry cleaning has many logical applications, but after close observation of going plants, the author is satisfied that a modern, well-regulated wet-washing job has a higher efficiency than a dry-table job, and for that reason wet washing must be used in difficult metallurgical operations where sulfur is the important factor.

Methods of Drying

As Forbes points out,[12] the sole advantage of dry cleaning is "avoidance of added expense on account of added moisture, either for transportation of the extra weight, or for evaporation in use." In view of this, attention is being given to drying the coal from wet washing.[13] Washed coal is dewatered by various methods, the principal ones being: dewatering elevators and screens; drainage bins and pits; centrifugal dryers, filters and presses; direct heat dryers.

The general scheme is (*a*) to dewater the top sizes above 1/2 or 5/8 in. by natural drainage, (*b*) centrifugally dry the −5/8 in. to 48 mesh and (*c*) dry the slimes (−48 mesh to 0) by heat, and thus obtain a wet-washed coal that is sufficiently dry for all ordinary uses.

Data as to what may be accomplished by natural drainage are difficult to obtain. One engineering concern that advocates a dual system says that "the moisture adhering to the washed coarse sizes is infinitesimal." Practical operating engineers are collecting data on dual systems and one made the statement to the author that the added moisture by draining the top sizes is 0.8 per cent.; other engineers find the added moisture from 1 to 2 per cent. by draining the top sizes and adding the dry fines.

Centrifugal dryers have been in use for several years. The latest development is the Carpenter dryer, used by the Colorado Fuel & Iron

---

[11] J. R. Campbell: *Op. cit.,* 773.

[12] W. A. Forbes: *Op. cit.*

[13] J. R. Campbell: *Op. cit.*

Co. This dryer has a rated capacity of 75 tons per hr. and will reduce the moisture to 5 or 5.5 per cent. if the fines are not excessive. The cost is not high (about $0.025 per ton) and is distributed as follows: depreciation (10 per cent.) $0.007; power, $0.010; maintenance, $0.008.

Heat drying of the sludge, or slimes, 48 mesh to 0, is a new development in the art and operating data are not available at present. The operating cost is estimated at from 5 to 8 c. per ton, depending upon the character of the fuel used, which in most cases may be a secondary coal recoverable from the refuse, but since the tonnage is small, usually not over 5 per cent. of the feed coal, the net cost is low.

A bituminous wet-washing plant now under construction in the Pittsburgh district will develop all the above methods of drying.

It is the feeling among coke-oven men that the extra heat required is not so important. This must be in view of the facts. To evaporate 1 lb. of water from and at 60° F. requires about 1100 B. t. u. To superheat the steam to standpipe temperatures raises this to about 2500 B.t.u. About 1200 B.t.u. are needed to coke 1 lb. of average coal in a by-product oven; 1 per cent. of moisture would require about 25 B.t.u.; 1 per cent. of moisture would replace 1 per cent. coal (requiring about 12 B.t.u.), so that the net change in heat requirement would be only ½ of 25 B.t.u.

What coke-oven men do object to is coal dripping wet, on the grounds of reduced oven capacity and possible effect on coke structure; objections which can largely be removed by modern methods of drying wet-washed coal. In certain other cases, good by-product practice demands a coal containing moisture, and sometimes water has to be added. For instance, swelling coals, which cause stickers and a deposit of carbon, demand a moisture content of 5 to 6 per cent.

Sludge Recovery and Water Clarification—Dust Recovery

The recovery of slimes is not the bugaboo it once was. Fines in water can be definitely collected by means of Dorr tanks and filters and then heat-dried. The underflow containing a 50-50 mixture of solids and water can be further dehydrated by filters to 15 to 20 moisture and the cake sent to direct heat dryers. The water is clarified for re-use.[14]

Dust recovery presents a more difficult problem. The methods employed are (*a*) cyclone dust collectors and (*b*) baghouse collectors. In going dry-table plants in the Pittsburgh and West Virginia districts equipped with cyclone dust collectors, the dust problem has not been completely solved. It is comparatively easy to get the dust out of the plant, but it goes elsewhere and in some cases becomes a nuisance to the inhabitants of the mining camp and to the farmers.

---

[14] J. R. Campbell: Mechanical Separation of Sulfur Minerals from Coal. *Trans.* (1920) **63**, 683.

The American Coal Cleaning Corpn., Welch, W. Va., has taken a long step to mitigate this evil by the use of baghouse dust collectors. This system is more expensive than cyclone dust collectors—about double the cost—but at the plant thus equipped at Berwind, W. Va., little or no dust was escaping through the bags. The plant had been operating several weeks and the bags were still white on the outside. It looked like almost a 100 per cent. job. A thin film of dust on the inside of the bags—about ⅛ in.—acted as a filter for the air.

Washing Efficiencies

One of the most important parts of a washing job is to determine the efficiency. We start the picture with a washability study of the coal to be cleaned, based on sink-and-float tests; therefore, we must logically end as we began. The "yardstick" by which to measure any washing job is sink-and-float tests and chemical analyses of the products. A 100 per cent. machine will remove all the impurities in the coal at the washing gravity with no loss of float coal. We all appreciate the fact that 100 per cent. is not expected of any washing apparatus, but certainly some approach the theoretical more closely than others.

Efficiency of washing based on sink-and-float tests is determined by Drakeley's formulas, which are as follows:

Qualitative Efficiency

$$100 \times \frac{\text{washed-coal float} - \text{feed-coal float}}{100 - \text{feed-coal float}}$$

Quantitative Efficiency

$$100 \times \frac{\text{feed-coal float} - \dfrac{\text{per cent. refuse} \times \text{refuse float}}{100}}{\text{feed-coal float}}$$

Using the following figures,

	Ash, Per Cent.	Float, 1.95 Sp. G., Per Cent.
Feed coal (100)	16.12	90.00
Washed coal (88.4 per cent.)	7.68	98.76
Refuse (11.6 per cent.)	80.20	1.76

by Drakeley's formulas we would obtain:

Qualitative Efficiency

$$\frac{100(98.76 - 90.00)}{100 - 90} = 87.6 \text{ per cent.}$$

Quantitative Efficiency

$$100 \times \frac{90 - \dfrac{11.63 \times 1.76}{100}}{90.0} = 99.76 \text{ per cent.}$$

Reduced to words: the qualitative efficiency is the amount of impurity removed from the coal at the washing gravity expressed on a percentage basis. Thus, if there is 10 per cent. impurities in the raw coal and the washer removes 8.5 per cent. the qualitative efficiency is 85 per cent. The quantitative efficiency is, strictly speaking, the amount of good coal recovered by the washer from the good coal in the feed. A rough approximation of the quantitative efficiency is to multiply the percentage of reject by the percentage of good coal in it and subtract the product from 100. A good many practical men simply use the multiplication and call

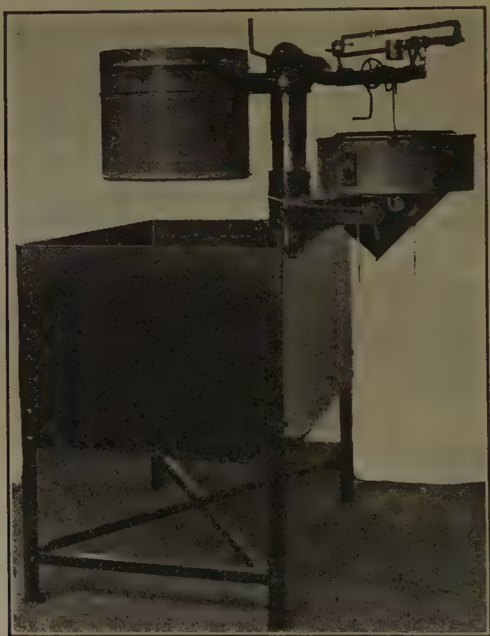


FIG. 13.—A DELATESTER.

the product "loss of feed coal" or "bank loss." This perhaps is the practical way of arriving at the cost of bank loss.

Some chemical engineers and practical operators believe that a standard of 85 per cent. for qualitative efficiency and 99.5 per cent. for quantitative efficiency, if the total reject is not excessive, is too high. The author is convinced that quality and recovery can be obtained simultaneously in a properly designed wet-washing system. In fact, the example cited is that of a going plant producing comparable results every day of operation.

For plant control, the author uses the Delatester (Fig. 13) because it is a standard machine for making sink-and-float tests in the plant or laboratory. After a sufficient number of these tests have been made to

CHEMICAL TEST DATA
on
__________ WASHERY
ASH

Qualitative Efficiency = _____ per cent. Quantitative Efficiency = _____ per cent.

Date	Feed Coal				Washed Coal				Refuse				Tonnage		Per Cent. Refuse		Hours	Sp. Gr.*
	Ash in Float	Ash in Sink	Comp. Ash	Per Cent. Sink	Ash in Float	Ash in Sink	Comp. Ash	Per Cent. Sink	Ash in Float	Ash in Sink	Comp. Ash	Per Cent. Sink	Coal	Refuse	By Wts.	By Calcn.		
Ave.........																		
Cum. Ave.																		

* Washing gravity and testing gravity must be the same.

Remarks:

FIG. 14.—CONVENIENT FORM FOR FLOAT-AND-SINK DATA.

determine the character of the products produced by the washer, chemical analyses may be relied upon to give sufficient control. All washery engineers understand this.

Only one gravity should be used in testing, and that is the gravity of washing. Some investigators use a dual gravity; that is, one gravity for washed coal and a lower gravity for testing the refuse. This practice is very misleading and leads to wrong conclusions. There may be conditions wherein it is advisable to use two gravities for testing both the sink in the washed coal and the float in the refuse. This would be permissible where there is a considerable percentage of middlings near the washing gravity. For example, if the washing gravity is 1.50 and there is considerable material between 1.40 and 1.60, a better picture of the washing results could perhaps be obtained by testing the washed coal and refuse at 1.40 and 1.60 instead of at the washing gravity of 1.50. Where a dual gravity is used, the results should be converted to a single gravity, that of the washing operation, before attempting to calculate efficiencies.

Fig. 14 shows a convenient form for keeping sink-and-float data.

Cost of Washing

The cost of washing may be divided into Capital Cost and Operating Cost.

The capital cost of any cleaning or preparation plant is dependent entirely upon conditions and the work demanded of it. Judging from the data at hand, an efficient wet-washing plant, complete with all accessories and housing, will cost from $400 to $800 per ton per hr. It is the common belief among operators that complete dry-table installations are considerably higher in first cost. Perhaps it will be fair to say that the air-table plant can be built for from $500 to $1000 per ton per hr., complete with all accessories and housing.

The operating cost may be conveniently subdivided into labor, power, maintenance, depreciation (and interest); loss of input, or "conversion cost."

Conversion cost is considerably more important than many are willing to understand. For example, 100 tons of run-of-mine coal costing $1.80 per ton is delivered to the cleaning plant. In going through the preparation plant there is a shrinkage of 10 tons, therefore there are only 90 tons of market coal that cost $180, or $2 per ton. In this case, the conversion cost is 20 c. per ton, a very large item in figuring the costs. It is more unfortunate if the 10 per cent. loss of input should contain a bank loss of 2 or 3 per cent. of good market coal. The sure way to get real money for coal is to get it into the railroad cars for market—not into the little refuse dump car that puts it where it is irretrievably lost. "Bank loss" is no misnomer, and is a real factor in washing costs.

Continuing on a percentage basis, the problem may be set up as follows:

90 per cent. = $1.80
1 per cent. = 0.02
3 per cent. = 0.06 (based on 30 per cent. coal in refuse)
7 per cent. = 0.14 (based on 70 per cent. rock in refuse)

Thus the 20 c. loss of input cost may be divided into 14 c. for real rock or slate with no value and 6 c. for good coal with a potential value of the full amount at least if shipped to market. Bank loss should be figured at more than the cost value, for if sold at a profit, the washing costs would be reduced. The anthracite operator probably is nearer right when he figures his bank loss at the sales value.

In the matter of the total cost of washing, the tendency on the part of many manufacturers is to "oversell" their equipment on this point. Coal cannot be washed for nothing, or near nothing, as many figure it. The splendid work of estimating engineers is a safeguard in this respect. The following are good average figures.

Cost of Washing per Ton (500 Tons per Hr. Capacity)

	Cents
1. Building upkeep	0.25
2. Machinery upkeep	1.40
3. Labor	3.00
4. Power	2.25
5. General overhead	1.70
6. Supplies	0.30
	8.90

These figures do not include depreciation and interest, which may add another cent or two, or "conversion cost," which is considerable in some cases if there is excessive "bank loss." Under ordinary conditions, coal can be washed in modern and efficient equipment for from 20 to 25 c. per ton including all items of cost.

The cost of washing can be predetermined very accurately from the known factors in the washing problem—the washability study and engineering data.

Effect of Washing

The immediate effect of coal washing, of course, is the reduction of ash and sulfur, both for commercial and metallurgical purposes. This can be translated into dollars and cents by the blast-furnace man if an improved coke is demanded by the exigencies of the case. The value of each 1 per cent. ash reduction in the coke ranges anywhere from 10 to 25c. per ton of pig iron, depending on conditions. Several large steel companies are studying this subject very intensively. Some figures

compiled by a fuel engineer showed that with a long freight haul, based on Sweetser's figures, 4 per cent. reduction of ash in the coal is worth $1 per ton to the consumer. Another blast-furnace man has a graph in which 1 per cent. of ash reduction in the coke is worth 40 c. per ton on pig iron on a certain furnace.

In inter-company business, it is almost certain that a substantial reduction of 3 or 4 per cent. ash in the coke will yield a good return on the cleaning equipment to the holding company and doubly assure a uniform and quality product of steel.

Another effect of washing, aside from the monetary consideration, is the improved physical structure of the coke. The cross-fractures caused by pieces of slate are eliminated, the general cell structure improved and blockier coke is produced. Even fine crushing, where intimate mixtures are made, does not eliminate the trouble caused by slate, which segregates in the coking mass and impairs the structure just as surely as the larger pieces.

In commercial washing, where the washed coal goes on a "buyers' market" in competition with an unwashed product, it sometimes is considerably more difficult to justify the cost of washing. The general buying public is apparently not yet educated to buying on a B. t. u. basis. It will be in time. A broadened and continuous market for a quality product will reflect a substantial saving in overhead costs—probably somewhat intangible, but there nevertheless. It means a six-day run instead of a four-day or a two-day run, and it costs money to run an idle mine.

Results of Washing on Going Plants

Forbes[15] shows some washing results which are undoubtedly based on washing the Pratt seam of coal in Alabama in a modified Elmore compartment jig. The figures of Table 2 are quoted from his paper.

As is well known in the South, this concern produces three products: washed coal, boiler coal and refuse. It will be noted that the secondary coal or middlings has practically the same fuel value as the raw coal. The coal losses are concentrated in the middling product, which makes the refuse clean. The production of a secondary coal is not always permissible, especially where there is no particular use for it, and for this reason we cannot build washers on such a basis. The coal washer in the North has to make two products—clean washed coal and clean refuse—which in some cases is very difficult to do.

The tabulation also indicates that the float in the refuse is apparently made on the dual basis, to which there is serious objection from a technical standpoint. The float is made at 1.37, whereas the washing gravity undoubtedly is considerably higher.

---

[15] W. A. Forbes: *Op. cit.*

Table 3 shows some results from plants covering the other types of apparatus mentioned in this paper, all based on metallurgical coal.

TABLE 2.—*Data of Washing Results**

Washer	Raw Coal				Washed Coal			
	Vol.	F. C.	Ash	Sulfur	Vol.	F. C.	Ash	Sulfur
No. 1	27.48	56.85	15.67	1.44	30.79	65.29	3.92	1.09
No. 2	27.61	62.25	10.14	1.43	29.13	66.23	4.64	1.17
No. 3	26.51	60.93	12.56	1.59	28.67	66.72	4.61	1.22
No. 4	25.34	61.46	13.20	1.86	27.77	67.71	4.52	1.28

Boiler Coal

Washer	Vol.	F. C.	Ash	Sulfur
No. 1	27.64	59.31	13.05	1.62
No. 2	26.00	59.80	14.20	2.00
No. 3	26.05	60.85	13.10	2.03
No. 4	25.12	61.20	13.68	2.20

Refuse

	Float at 1.37			Sink at 1.37		
	Per Cent.	Ash	Sulfur	Per Cent.	Ash	Sulfur
No. 1	3.9	3.85	1.10	96.1	70.62	2.14
No. 2	4.2	4.79	1.27	95.8	61.83	4.67
No. 3	4.9	4.92	1.35	95.1	63.33	4.08
No. 4	4.7	4.70	1.48	95.3	56.00	5.24

* Figures taken from paper by Forbes.

TABLE 3.—*Results with Various Methods and Metallurgical Coals*

	Table Plant *A*		Table Plant *B*		Jig and Table Plant *A*		Jig and Table Plant *B*	
	Ash, Per Cent.	Sulfur, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.
Feed coal	13.00	2.50	11.00	2.00	12.00	2.80	9.80	1.66
Washed coal	8.80	1.30	6.50	0.90	8.20	2.00	8.20	1.37
Refuse	47.0	12.20	29.60	4.65	45.00	11.11	43.40	7.90
Reject, per cent.	11.0		20.00		9.00		4.5	
Seam of coal	Kittannings		Freeport		Illinois No. 6		Pittsburgh No. 8	

TABLE 4.—*Results with Rheolaveur Washer on Metallurgical Coal*

Retained on	Weight, Per Cent.	Cum. Weight, Per Cent.	Ash, Per Cent.
5⁄16 in. or 2½ mesh	0.7	0.7	33.7
⅛ in. or 6 mesh	23.3	24.0	23.2
1⁄16 in. or 14 mesh	39.2	63.2	15.5
48 mesh	25.4	88.6	17.0
100 mesh	6.0	94.6	18.8
200 mesh	2.6	97.2	20.0
Through 200 mesh	2.8	100.0	18.1

AVERAGE 17 PER CENT. ASH FEED

1927	Figured Recovery, Per Cent.	Washed Coal Ash		Refuse Ash		Actual Recovery, Per Cent.
		Actual Per Cent.	Theoretical Per Cent.	Actual Per Cent.	Theoretical Per Cent.	
Jan. 12 to 31	82.1	10.20	9.0	48.10	51.1	
Feb. 1 to 15	82.6	10.10	9.1	49.70	51.9	
Feb. 16 to 28	83.2	10.15	9.4	51.20	53.0	
Mar. 1 to 31	83.2	10.17	9.4	50.83	53.0	81.8
July 1 to 31	82.0	10.30	9.8	49.20	51.0	82.6

The results on the Rheolaveur washer (Table 4) on metallurgical coal are shown somewhat more in detail, since such figures are available. The figures are taken from Spahr's paper.[16] A screen analysis of the washed coal shows that the 14-mesh to 48-mesh material is washed prac-

TABLE 5.—*Data of One Day's Run from Dry-cleaning Plant*

Coal	Size	Clean Coal Ash, Per Cent.
Nut	2 to 1 in.	9.15 domestic size, bone left in
Pea	1 to ½ in.	7.26 domestic size, bone left in
Slack	½ to ¼ in.	6.62 domestic size, bone left in
	¼ to ⅛ in.	6.23 domestic size, bone left in
	⅛ to 0 in.	7.21 domestic size, bone left in

	Ash, Per Cent.
Average 2 to 0 in. clean coal with bone left in, nut and pea sizes	7.21
Average 1 to 0 in. clean coal with bone left in, 1 to ½ in. size	7.08
Average ½ to 0 in. slack	7.07
Theoretical ash in coal	5.50
Theoretical ash in refuse	86.00
Refuse, 6 per cent. of whole	80.00

Crude 2 to 0 in. without middlings but with all bone pickings of sizes over 2 in. crushed through 2-in. single roll crusher added 10.98 per cent. ash.

[16] C. Spahr: Washing Bituminous Coal with the Rheolaveur Process at the Cokedale Plant of the American Smelting & Refining Co. Rocky Mountain Coal Min. Inst. (1927) 49.

tically as well as the larger sizes. These results are somewhat different from other gravity methods.

It has not been possible to obtain extensive figures from dry-cleaning plants. The results shown in Table 5 are from a going plant and were taken throughout the entire day of Aug. 11, 1927. It is the intention at this plant to recover in the domestic sizes the bone coal which has been crushed from the pickings of the lump and egg sizes.

No data have been given on bank loss. The analyses of the refuse will give some indication of this loss to the reader who is acquainted with the characteristics of the coal and who knows what the ash in the refuse ought to be at the washing gravity. It was not possible to get sink-and-float data in all cases. There is perhaps a wide range of bank loss anywhere from 5 per cent. of float coal in refuse from the best performance to 40 per cent. of float coal in refuse from the worst performance. In some cases, not listed here, washery wastes have shown 30 to 40 per cent. float coal at the washing gravity.

Conclusion

The aim of this paper has been to emphasize the following fundamentals in the mechanical cleaning of bituminous coal:

1. Necessity of washability studies.
2. Type of plant adaptable to coal studied.
3. Plant control and performance.
4. Cost of washing—all factors.

The new era demands that the coal-preparation equipment be just as efficient as the modern by-product coke oven and blast furnace. If a coal demands wet washing, it should be wet washed, but it should be wet washed well; if it demands dry cleaning, it should be dry cleaned, but that also should be well done—for, to quote the old adage, "whatever is worth doing at all is worth doing well."

Acknowledgments

The author acknowledges cooperation and help from the following: John Griffen and George S. Scott of the American Rheolaveur Corpn., without whose painstaking efforts in assembling data the paper could not have been put into its present form; C. A. Meissner of the United States Steel Corpn., for advice in outlining the paper; H. J. Rose of the Koppers Co., W. J. O'Toole of the American Coal Cleaning Co., Lee Llewellyn of the Pittsburgh Coal Washery Co., A. J. Sayers of the Link Belt Co., Ray W. Arms of Roberts & Schaeffer Co., Emil Deister of Deister Machine Co., R. T. Bliss of Deister Concentrator Co., Walter L. Remick, the Hydrotator Co., and many engineers and operators in the large coal and steel companies for data and advice on operating problems, costs, and so forth.

DISCUSSION

C. A. Meissner, New York, N. Y.—The blast-furnace people are interested in the cleaning of coal, for clean coal gives us clean coke. To the average coal man, naturally, it means extra work.

What I like about Mr. Campbell's paper is the thoroughness with which he has gone into the matter, and the fairness, taking up the different subjects of coal washing and the different methods of coal washing and coal cleaning, wet and dry.

I have always had the feeling that I wanted the dry method. So far, we have not succeeded on a large scale. There has been good work done in dry cleaning in the country but the disadvantage of dry cleaning has been the necessity of sizing all the coal definitely.

The most interesting part to me is his study of the washability of coal. From those studies, if they are interested, blast-furnace operators can determine fairly well whether the coal that is used for their coke can be washed economically and satisfactorily. That has not been brought out before so clearly in any paper we have had, and it is valuable.

We need more coal washing, more coal cleaning. We have had experience, for instance, with some of our coals when by better and more careful mining, by taking out the slate and getting the coal clean, we have materially improved the quality of the coke. It is not a question of taking the coal out of the ground and making the best of it; it is a question of taking it out of the ground in the best manner.

We will come to coal cleaning more and more. The demands made upon the blast-furnace men by customers are becoming severe. Competition is becoming more severe, and it is surprising how much can be done in the way of more economical work and better quality, more uniform quality of pig iron, if we have a uniform coal. We can get that coal, as a rule, by cleaning. There are mines that are delivering coal that does not have to be cleaned, but I have in mind mines, which are doing good work, that run in ash, for instance, from 6 to 14 or 15 per cent. carload lots. That means irregularity, and that is where the advantage of cleaning comes in.

R. E. Rightmire, Fairmont, W. Va. (written discussion).—This paper is comprehensive as to methods of investigation and processes for mechanical cleaning.

In carrying out float-and-sink tests, which are universally recognized as basic in such investigations, there appears to be difference of opinion as to the practicability of attempting separation of the smaller sizes; some investigators report considerable difficulty in getting dependable separation of sizes below 20 mesh, while the author advocates separation down to 48 mesh for results in calculating composites.

Each of the two outstanding processes, dry and wet, has its advantages and disadvantages, and final selection undoubtedly should be determined largely by the nature of the coal itself as a cleaning problem, together with the standard demand of the finished product.

The moisture in wet-cleaned coal is an important feature, especially for such coal in commercial shipments. Aside from its effect on quality and ultimate cost to the consumer is the important consideration of frozen contents of railroad cars when the wet-cleaned coal moves in the colder climates.

The cost of reducing satisfactorily such moisture content is a variable, dependent principally, as Mr. Campbell points out, on cost of power and for maximum reduction on the additional cost of heat for drying.

Dry cleaning is free from the moisture difficulty but it presents instead the dust problem, which apparently has not been satisfactorily solved to date. For greater advantage and more favorable reception to cleaning installations undoubtedly it will be necessary to work out some less expensive method than is yet extant for dust elimination as well as moisture reduction.

The need for mechanical cleaning is dependent on inability to secure by some other, less expensive, practical methods the quality of coal necessary, for, as Mr. Campbell has pointed out, we cannot expect to wash coal without appreciable cost and we cannot adopt any unusual measures in handling coal for betterment in quality without some addition to the ordinary cost.

Frequently the ash content under regular production methods is satisfactory but sulfur content due to mine variations is more difficult to control. Such conditions do not necessarily call for mechanical cleaning to produce satisfactory product, for in many instances selection according to sulfur content in individual working places and control of loading for shipments gives a satisfactory product and renders unnecessary the expenditure for cleaning-plant installation and attendant operating costs.

It would seem that mechanical cleaning will come into increasing favor with the more general adoption of mechanical loading of coal in the mines, but there are some nice problems to be worked out in that connection and some careful balances to be struck before the coal producer can be assured positively that the mechanical cleaning of coal mechanically loaded in the mine is the most advantageous road to steady, satisfied markets and satisfactory money return on his investment and efforts.

C. E. Locke, Cambridge, Mass. (written discussion).—Mr. Campbell's paper gives added evidence that after a long period of comparative inaction the operators are coming to realize the need of removing impurities from coal and especially of studying how the removal can be made most cheaply and efficiently.

As compared to the concentration of ores, the cleaning of coal has progressed more slowly from the viewpoints of both theory and practice. The fault for this does not lie entirely with the operators, because the economic conditions in the two industries are quite different. Coal is a relatively low-priced commodity found in abundance in our country. Competition is keen and the margin of profit is small. The general custom of marketing coal on a quantity basis (by tonnage alone) rather than on a quality basis (by determination of British thermal units, ash and sulfur) still prevails at most mines. Consequently, there has not been the incentive to produce clean coal. In fact, the result has been just the opposite. The main thing desired has been tonnage and as long as a coal company could sell a carload of coal containing a quantity of removable slate at exactly the same price per ton as could be obtained for clean coal after the slate was removed, why should the company go to the expense of removing the slate, and at the same time lose the money that would have been received for this same slate if it had remained mixed with the coal?

Perhaps of all the ores, iron ore comes nearest to coal in being relatively cheap and abundant, but even that is not handicapped by the quantity or "tonnage" factor. Washing operations on the Mesabi iron range are governed largely by the call for high-grade product rather than high extraction of iron. The amount of iron lost in the tailings from washing would surprise a coal man who would feel that he was a profligate if he let good coal go to waste in the same proportion.

Mr. Campbell, perhaps unconsciously, has given an excellent illustration of the predominance of the idea of tonnage in the minds of coal men. In discussing costs on pages 328 and 330, he includes "bank loss," or loss of coal in the cleaning operation as an item of cost. In other words, costs are calculated per ton of cleaned coal whereas on ores the custom is to figure all costs to the basis of a ton of raw material before treatment, and loss of values during treatment is a deduction from the original gross value of his ore. Actually, the end point in the two cases is the same, but the viewpoints are very different. The ore man is aiming for quality and the higher the grade of his concentrates, the better the price he gets, and he is often warranted in throwing away values in the waste product in order to attain his end. It is stated that Marcus Daly, when told that he was throwing away money in the form of the large amount

of copper that he was allowing to go to waste in the tailings of his Anaconda mill, replied to the effect that this did not worry him as copper was the cheapest thing he had. The viewpoint of a coal man makes it difficult for him to see that the "quality" factor often makes it possible for an ore man to net more profit by rejecting 10 per cent. of his values with the waste than by rejecting only 5 per cent., and that sometimes without any concentration no profit at all may result.

In all concentrating operations there are the three elements of capacity, efficiency and cost which are more or less opposed to one another. Higher capacity means lower efficiency with lower cost per ton, but may in the end yield a higher daily profit.

Mr. Campbell aptly remarks that the general public is not yet educated to buying on a quality basis, but adds hopefully that it will come in time. Let us speed the day! For when that day comes, it will mean the solution of some of the present financial problems in coal cleaning. But in the meantime, not only the public but also the operators can gain by education. There is need of closer cooperation between ore dressing and coal washing for exchange of ideas. Both parties have kept too much by themselves. It is significant that when Mr. Campbell's paper was presented it did not even come to the coal session, but was read at the iron and steel session and that coal-washing papers were segregated from ore-dressing papers, which were handled by the Milling Committee.

Leaving the economic side, the fundamental principles and processes for concentrating ores and coal are exactly the same. The investigations of Robert H. Richards and others on ores have demonstrated the laws of sizing, of classification by free and hindered settling, of jigging with pulsion and suction, and of washing on tables. An elementary knowledge of these laws is sufficient to show the advantage of jigging a sized product as compared to an unsized product, or of using classified products as feed to tables as against unclassified feed, and the futility of expecting a coal washer of the classifier type, such as the Robinson washer or the Menzies Hydro-Separator, to be able to remove anything but the coarser particles of slate, leaving the fine still mixed with the coal. There still remains the unexplored field of stratification, as in the Rheolaveur, the laws of which have not been studied in an experimental way. When these laws become known, it is not at all unlikely that they will be found to have application in the treatment of ores as well as coal.

It is gratifying to find that for this country there seems to be coming a realization of the value of washability curves, as illustrated by Mr. Campbell, in interpreting the results of float-and-sink tests. We adopted the float-and-sink apparatus for testing years ago, but have lagged behind Europe in methods of plotting and studying results.

The use of the word "mesh" in designating size of opening in screens does not give the reader the exact size of the hole. Perhaps Tyler Standard screens were used, although nowhere is it so stated. Incidentally, here is a good opportunity for the coal man to educate the ore man instead of following his lead. Unfortunately, on ores custom has sanctioned the use of the term "mesh" although it is meaningless as far as designating the exact size of the screen hole is concerned. With coal, especially anthracite, screen openings have been given exactly in fractions of an inch. Now is the time for the coal people to rally to a call for standard use of either inches or millimeters and do away with the indefinite term mesh. Would it be too much to ask Mr. Campbell to set a good example by changing his mesh to exact sizes in millimeters?

J. R. Campbell (written discussion).—Mr. Meissner speaks about the thoroughness and fairness with which the subject has been handled. There is no other way and it is always the aim of the writer in treating technical subjects to "hew to the line." There are some outstanding coal-washing jobs on the market and there are others, as Professor Locke says, that have limited application. The coal operator

must make his own choice and live with his job after he has it. The writer believes in educational work of the broadest kind and, if the purveyor of coal-cleaning equipment has a good article, the coal operator will find out for himself whether or not it fits his conditions.

Regarding Mr. Rightmire's remarks on washability of 48-mesh material, the writer wishes to emphasize the point that 20 mesh is no longer the limit in modern wet-washing equipment, although 20 mesh *was* the standard for a good many years. This point is most important in wet-washing coal for metallurgical purposes where it is already friable and where it has to be crushed to ⅜ in. or under to liberate the impurities—probably with the production of excessive fines.

As to the word "mesh" used all through the paper, Tyler standard screens are used for the square meshes and round hole openings for the large sizes. The square mesh work starts at 14 mesh.

The writer agrees thoroughly with Professor Locke that standards ought to be adopted and used, giving the mesh openings either in inches or millimeters. It is too late to set the good example asked for by Professor Locke.

Test for Measuring the Agglutinating Power of Coal*

By S. M. Marshall,† New York, N. Y. and B. M. Bird,‡ Tuscaloosa, Ala.

(New York Meeting, February, 1929)

For a number of years European investigators have used laboratory methods of predicting the probable strength of coke made from coal, and recently several investigators in the United States have reported the use of various tests for this purpose. There is at present no method accepted as standard, however, nor is there even agreement as to the general type of test that should be used. Furthermore, because the details of most of the tests have not been worked out carefully, one investigator can not hope to duplicate the work of another.

The extension of coke manufacture to new localities in the United States, and the increasing need for cokes of exceptional qualities made from the most economical mixtures, made it desirable to have some simple laboratory test which will determine whether given coals, alone or blended with other coals, are more satisfactory from the standpoint of coke strength than others of different cost. The work described herein has resulted in the development of a simple laboratory test which, it is believed, can be generally adopted in coal laboratories, and which will give concordant results even though performed by different investigators in different laboratories.

The general method of predicting whether a coal or a mixture of coals will produce a strong coke, resistant to crushing and shattering, has been to correlate the known strengths of the cokes produced in commercial ovens with some measurable constituent or with some property of the coals, and then to use the relationship so found for forming an estimate of the coking strengths of coals unknown. Several such correlations have utilized the percentages of the various ultimate constituents of coal, such as the oxygen content and the hydrogen-oxygen ratio;(39)§ other correlations have involved the proportion of α, β and γ compounds,(17,21,30a) or of "oil and solid bitumens."(16,30) Also a relationship has been traced between the coking properties of coal and the rate of change of the resistance offered to the flow of an inert gas by fine particles of the coal while being slowly heated.(11,17b) But the most general corre-

* Published by permission of the Director, U. S. Bureau of Mines.

† Consulting Engineer (Perin and Marshall).

‡ Supervising Engineer, Southern Experiment Station, U. S. Bureau of Mines; at time of investigations described herein, Supervising Engineer, Northwest Experiment Station, Seattle, Wash.

§ Numbers in parentheses refer to bibliography at end of paper.

lation has been made between the properties of the coke and the agglutinating power of the coal as determined by some test involving the capacity of the coal to cohere large proportions of some inert[1] material, such as sand, or a pulverized carbonaceous material.

The agglutinating test to be described in this paper was developed during an investigation conducted by the U. S. Bureau of Mines, the Pacific Coke & Coal Co. and the University of Washington. During this investigation, which had for its purpose determination of the washing and coking properties of certain coal seams in Pierce County, Wash., the senior author was faced with the problem of determining whether cokes manufactured from those coals would be suitable for use in the iron blast furnace. Of prime importance was the question of whether the cokes would have the required strength and the required resistance to shattering. As there was no blast-furnace plant in that territory, and as only small samples of the coals were available, resort was naturally had to existing methods of predicting the strength of cokes by agglutinating tests, such as those first proposed many years ago by Richters(35) and Campredon.(8) However, a careful study of these and subsequent tests revealed many particulars wherein the procedures seemed faulty. The effort to evolve a more reliable test resulted in one containing the essential features of that to be described in this paper.

After concordant results could be obtained on duplicate tests of the same coal the test was used to determine the agglutinating values of coals now used in various parts of the United States in the manufacture of coke for the iron blast furnace. Coals were selected that give strong cokes, medium strong cokes and weak cokes. Between the known strengths of these cokes and the agglutinating values of the coals a very consistent relationship was found, indicating that the test affords a reliable index to the coking strength of coals. For this reason it seemed to merit consideration as a standard method. However, because many parts of the procedure in the original investigation had been worked out hurriedly it was felt that before the test was offered as a standard method those details should receive further study. This supplementary investigation has been carried on at the Northwest Experiment Station by the junior author and Kenneth A. Johnson, junior chemist.

As the procedure in its present form is usable and reliable, this paper has been prepared as a progress report of the work for the benefit of those interested in the coking of coal.

Summary of Previous Tests

Practically all agglutinating tests involve the carbonization under standard conditions of a small sample of carefully prepared coal, either

[1] This term is used throughout the paper although it is recognized that none of the so-called inert materials are truly inert in agglutinating tests.

TABLE 1.—*Various Agglutinating Tests of Coal*

Authors	Reference in Bibliography	Inert Material			Preparation of Coal
		Nature	Mesh	Preparation	
Richters (1870)	35	Quartz	Powdered form	Finely washed	Air-dried finely ground
Campredon (1895)	8	Sand—rounded grains	−25(100 per sq. cm.) +50(400 per sq. cm.) mesh		Put through 50 mesh, dried 100° C.
Pollard (1908)	32	Sand—rounded grains	−40 + 50 mesh		Put through 50 mesh
Lessing (1912)	24, 10		Inert material not used		Air-dried coal put carefully through 30 mesh I. M. M. to avoid fines
Dunn (1913)	15	Anthracite coal	−1 mm. + 0.5 mm.		Put through 0.5 mm. sieve
Meurice (1914)	29, 13	Sand—rounded grains	−40 + 50 mesh	Washed 20 per cent. sol. HCl	Put through No. 80 sieve; dried at 100° C.
Charpy and Godchot (1917)	9		Inert material not used		Put carefully through 120 mesh to avoid fines; kept in saturated air several days at 18°–20° C.
Sinnatt and Grounds (1920)	37	Electrode carbon	Varying meshes, sized material		Put through 1/60 mesh
Weighell (1921)	38	Electrode carbon	−100 + 120 mesh		Put through 60 mesh
Bone, Pearson, Sinkinson and Stockings (1922)	6	Dry coke	−30 mesh	Made at 900° C.	Used ingredients extracted from coal, as well as coal
Gray (1923)	19, 28	Sand—sharp-edged grains	−40 + 50 I. M. M.		Put through 50 mesh I. M. M.
Badarau and Tideswell (1923)	2, 7	Electrode carbon	−60 + 90 I. M. M.		Put through 60 mesh
Ahrens (1924)	1, 23a	Dry coke	−B50 mesh	Coke prepared by carbonizing same coal under conditions of test	Preserved under water; well dried; screened through B30, on B50 mesh
Qvarfort (Apr., 1925)	33		Inert material not used		Screened through 35 on 45 mesh
Marson and Cobb (July, 1925)	25	Various ash-forming constituents	−100 mesh		Put through 20 mesh I. M. M.
Barash (Nov., 1925)	3, 12, 18	Electrode carbon	−60 + 90 I. M. M.		Put through 90 mesh (I. M. M.) by degrees to avoid fines; kept in sealed containers
Kattwinkel (1926)	22	Sand—rounded grains (?)	−50 mesh	Ignited	Put through 50 mesh
Coffman and Layng (1928)	11		Inert material not used		Screened through 20 on 60 mesh

TABLE 1.—(*Continued*)

Authors	Preparation and Testing of Buttons				Source of Heat
	Weight	Miscellaneous Preliminaries	Container	Temperature of Coking	
Richters (1870)	1 g. of coal + varying wts. of quartz powder		Platinum crucible ht. 30 mm. Dia. at top (?) 30 mm.	Approx. 950° C	Burner
Campredon (1895)	1 g. of coal + varying wts. of sand		Volatile crucible	Bright red	Muffle (?)
Pollard (1908)	25 g. of varying proportions of coal and sand		Platinum crucible	950°C. (?)	Burner (?)
Lessing (1912)	1 g. coal		Quartz glass tubes (10 mm. dia.) with quartz piston resting on charge	600° and 900° C.	Electric tube furnace
Dunn (1913)	10 g. of varying proportions of coal and inerts		Porcelain crucible	Bright red (?)	Burner
Meurice (1914)	1 g. of coal + 17 g. of sand	Level surface of mixture	Porcelain crucible; ht. 3.5 cm. dia. 4.0 cm. at top	800°–900° C.	Muffle
Charpy and Godchot (1917)	Entirely filled tube	Coal packed in tube and compressed under 5 kg. per sq. cm.	Refractory cylinder (12 mm. dia. 20 mm. ht.). Closed on ends with sheet-iron disks.	Varied 650°–1000° C.	Electric muffle
Sinnatt and Grounds (1920)	1 g. of varying proportions of coal and inerts		Volatile crucible	950° C.	Bunsen burner
Weighell (1921)	5 g. of varying proportions of coal and inerts		Platinum crucible	950° C. (?)	Burner
Bone, Pearson, Sinkinson and Stockings (1922).	9 g. coke dust + 1 g. extract	Coke dust saturated with extract or ground with it in mortar	Cylinder 1 in. dia. (inside)	850°–900° C.	Electric muffle
Gray (1923)	25 g. of varying proportions of coal and sand		Platinum crucible dia. at base 24 mm., dia. at top 34 mm. ht. 40 mm.	1000° C.	Burner
Badarau and Tideswell (1923)	1 g. of varying proportions of coal and inerts		Silica crucible ht. 1.5 in.	900° C.	Burner
Ahrens (1924)	2 g. of mixture of 50, 60, 70, 80, and 90 % of coke	Mixture shaken down and leveled	Porcelain ring, 16 mm., dia. × 16 mm. ht., with base and cover	650°–700° C.	Electric muffle
Qvarfort (Apr., 1925)	6 g. of coal	Crucible tapped gently	Conical quartz crucible; length 220 mm. upper dia. 14 mm. lower 10.5 (inside)	Cold furnace to 1000° C.	Electric tube furnace
Marson and Cobb (July, 1925)	Approx. 300 g. of 95 per cent. coal, 5 per cent. inerts	Tapped down	Fire clay tubes, 1⅞ in. dia. × 12 in. length (inside)	500° and 800° C. start cold furnace	Muffle
Barash (Nov., 1925)	1 g. of varying proportions of coal and inerts	2-g. lots mixed together. Very thorough mixing in dry state.	Silica tube 2 in. length ½ in. dia. (inside)	800° C.	Electric tube furnace
Kattwinkel (1926)	1 g. coal + 10 g. sand		Platinum crucible	800°–900° C. (?)	Burner
Coffman and Layng (1928)	Column of coal 10 cm. long	Tube tapped	Pyrex tube 14 mm. dia.	0°–600° slow rise	Electric tube furnace

TABLE 1.—(*Continued*)

Authors	Preparation and Testing of Buttons		Agglutinating Value	Remarks
	Time of Coking	Test of Coke Buttons		
Richters (1870)	Until flame disappears	Must just sustain 500 g. wt.	Ratio: $\frac{\text{Wt. of quartz}}{\text{Wt. of coal}}$	Amount of quartz powder is varied by 0.1 g. until button is just strong enough to withstand ½ kg. pressure.
Campredon (1895)	Until flame disappears (?)	Coherency determined by visual examination	Maximum ratio: $\frac{\text{Wt. of sand}}{\text{Wt. of coal}}$	
Pol ard (1908)	7 min. (?)	Coherent button that just fails under 500 g. wt.	Maximum ratio: $\frac{\text{Wt. of sand}}{\text{Wt. of coal}}$	
Lessing (1912)	15 min. at 600° C. or 7 min. at 900° C.	Appearance of buttons	Coals classified by comparing buttons with those of coals of known characteristics	Runs laboratory by-product test in making buttons.
Dunn (1913)	Until flame disappears (?)	Coherent button that fails under slight pressure	Maximum ratio: $\frac{\text{Wt. of inert}}{\text{Wt. of coal}}$ (?)	Objects to sand because of partial fluxing with ash in coal.
Meurice (1914)	Until flame disappears	Crushing strength determined: not over 1 g. of detritus	Ratio: $\frac{S \times C}{P}$ where "S" = Wt. sand (17 g.) C = Wt. to crush button (Kg.) P = powder (g.)	
Charpy and Godchot (1917)	1 hr.	Crushing	Crushing strength kg. per sq. cm., average of 6 determinations	Experiments showed variations in temperature of coking to have marked effects on agglutinating value.
Sinnatt and Grounds (1920)	7 min.	Coherent button that just fails under 100 g. wt.	Curve of average mesh of inert material plotted against maximum ratio: $\frac{\text{Wt. of inert}}{\text{Wt. of coal}}$	Suggested fineness of inert material just destroying caking properties be taken as measure of agglutinating value.
Weighell (1921)	7 min.	Coherent button that just fails under 100 g. wt.	Maximum ratio: $\frac{\text{Wt. of inert}}{\text{Wt. of coal}}$	

Bone, Pearson, Sinkinson and Stockings (1922).	6–8 min.	Crushed in testing machine	Crushing strength, lb. per sq. in.	100 g. wt. on button during coking. Buttons ground to cylinders 18 mm. high before crushing.
Gray (1923)	7 min.	Must just sustain 500 g. wt. detritus less than 5 per cent.	Maximum ratio: $\frac{\text{Wt. of sand}}{\text{Wt. of coal}}$	Correlated his agglutinating values with quality of coke produced by coking samples of coal in commercial ovens.
Badarau and Tideswell (1923)	7 min.	Crushed in testing machine	Curves crushing strength in kg. against ratio: $\frac{\text{Wt. of inert}}{\text{Wt. of coal}}$	
Ahrens (1924)	10 min.	Smoothed on top; crushed in testing machine	Composition last coherent mixture, and strength of button. Average 5 buttons; value deviating greatly ignored	Stresses need of uniform volume in coke buttons, plots results in form of curves besides using definite values.
Qvarfort (Apr., 1925)	20 min.—0° to 1000° C. 50 min. at 1000° C. 1 hr. cool part of furnace	Small sections of cylinder crushed in motor-driven machine. Various sections examined visually	Sum of 4 numerical factors, appearance of "pressure cone," of sectional surfaces, the crushing strength and coke yield, represents the quality of the coke	Crucible raised and lowered mechanically through hottest zone of furnace during heating.
Marson and Cobb (July, 1925)	2 hr. and 3 hr.	6 to 10½-in. cubes cut from coke; trued-up; crushed in testing machine	Average crushing strength of cubes lb. per sq. in.	Though not presented by authors as an agglutinating test, this test does give a direct measure of coke strength.
Barash (Nov., 1925)	5 min.	Button must just fall completely to powder	Minimum ratio: $\frac{\text{Wt. of coal}}{\text{Wt. of inert}}$ Average 2 runs	
Kattwinkel (1926)	Until flame disappears (?)	Crushed in testing machine with load added at uniform rate	Meurice ratio	
Coffman and Layng (1928)	Variable, about 1½ hr.	Passing nitrogen gas through coal while heating	The maximum value of $\frac{dp}{dt}$ or of the ratio of the increment of pressure in mm. of water to the increment of time in min.	The pressure is a measure of the resistance offered by the coal gradually heated to the passage of nitrogen gas.

alone or mixed with an inert material, and some tests of the resulting coke button which serve to indicate the agglutinating value of the coal. To facilitate comparison and discussion of these tests Table 1 has been prepared, giving the essentials of each. The table has been divided into columns showing (1) whether or not an inert material is used and, if so, its nature and preparation; (2) the method of preparing the coal; (3) the details of preparing and testing the buttons; and (4) the agglutinating value The intention has been to arrange the different tests in the order in which they were first proposed but in detailing them to use the latest published description. Whenever several investigators have used essentially the same procedure, a reference has been made to the fact in the appropriate column.

1. The use or nonuse of an inert material is one distinguishing feature of any test. Some form is used in all but four of the tests. In the earliest tests the material employed was sand, usually composed of rounded grains; in the more recent tests electrode carbon has been favored. Most of the tests using an inert material specify that it shall be closely screened and in some instances that it receive other preparation.

2. In the preparation of the coal no two tests are alike. The sizes of screens through which the coal is crushed vary from 30 to 120 mesh; three tests even involve the use of sized coal. Everything from coal dried at 105° C. to coal saturated with moisture has been used; often there are no specifications as to moisture content.

3. Equally great diversity of opinion is shown in regard to the best methods of preparing the coke buttons. In some tests both the weight and the volume of the buttons before coking are varied; this group includes all tests in which varying weights of inert material are added to a constant weight of coal. In others the volume only is varied; this group includes tests in which the weight of button is kept constant but the proportions of coal and inert material are varied. Only three tests involving the use of an inert material maintain both weight and volume constant. In two of the four tests using no inert material volume alone determines the size of buttons. Under the caption in Table 1—Miscellaneous Preliminaries—are included what are probably the weakest points of most procedures for the preparation of the buttons. No description of any test involving the use of inert materials records adequate precautions for preventing segregation of the ingredients of the buttons in the interval between mixing and coking. Only one investigator seems to have recognized the necessity for compressing the coal, or the coal and inert mixtures, before coking in order to bring the components of the buttons into intimate contact. The container for the buttons generally adopted in the earliest tests was the crucible, but in recent procedures the tendency has been to use cylindrical tubes. The temperatures of coking are generally in the range from 800° to 1000° C.;

in three tests the coal is heated slowly. As sources of heat the muffle and the Meker or Bunsen burners are favorites in the early tests; in the most recent some form of special tube furnace is generally recommended. The time of coking, if fixed, is generally 5 to 7 min., but commonly it is a variable period depending upon the evolution of volatile matter.

The tests of buttons are mainly of two kinds. Those buttons in which the proportions of inert material are varied must generally withstand some fixed test such as supporting a given weight, sometimes with a requirement as to the percentage of detritus; several tests depend upon visual examination or upon some other scheme involving the skill and judgment of the operator, such as the breaking of the button in the fingers. Buttons made with constant proportions of coal and inert materials are generally crushed in a testing machine. Those containing only coal are tested by some means suited to other conditions of the test. Only three tests involve truing-up the surface of the buttons prior to crushing. The test of Coffman and Layng,[11] which is an application of Foxwell's "plastic curves"[17a] to the measurement of agglutinating values, consists of passing an inert gas through the coal while the carbonization progresses.

4. The favorite agglutinating value has been the maximum ratio: Weight of inert material divided by weight of coal in the button, which just met or failed to meet the breaking test that the investigator adopted. In some tests this has a single value, the maximum attained; in others, the results are shown as a curve in which the breaking strength is plotted against the ratio of inert material to coal. In a few cases an equation is used in which the inert-material ratio, breaking load, percentage of fines produced by crushing, and/or other factors are incorporated to produce a factor which is merely a number that has no direct relation to the load carried by the button. In several tests the crushing strength per unit area is the agglutinating value.

Choice of Procedure for Agglutinating Test

An examination of the work of other investigators leads to the conclusion that the general type of agglutinating test involving the coking of small amounts of coal and the testing of the resulting coke buttons is sound in principle, but that the procedures of the tests thus far proposed are faulty. Accordingly, all effort during both the original and the supplementary investigation has been concentrated on working out the details of the type of test of which the fundamentals have already been shown to be satisfactory. Each step in the procedure has been studied, and if some part of a previous test has met the requirements, it has been used; only when necessary has a totally new procedure been

developed. The proposed procedure will be discussed[2] under the same divisions as made in Table 1.

Inert Material

In the choice of an inert material the important considerations are suitability of the surface for deposit of the agglutinant and practicability of preparing duplicate quantities of the inert material. For providing a surface on which the agglutinant will deposit sand, coke, anthracite coal and electrode carbon have been shown by various investigators to be satisfactory. The deciding factor as to which of these to use in a standard test is the practicability of preparing duplicate samples of precisely the same quality. In this connection Barash[(3)] reports ". . . using the same samples of coal and the same method but different samples of the inert material, electrode carbon, that identical results were obtained." Fulweiler and Cleveland[(18)] also report concordant results in changing from electric arc-carbon pencils to Acheson graphite, but that they obtained higher values in changing to high-carbon coke. Further they report, although using slightly different sizes of sieves for screening the electrode carbon, that they succeeded in obtaining concordant results on two of the coals tested previously by Barash. These results are surprising. Certainly, judging from the nature of this material and its mode of preparation, no one could reasonably expect to prepare at different times samples of electrode carbon that would be more than approximately alike. One would expect the shape of grain and the porosity of the individual grains, which has been shown to affect the agglutinating values vitally,[(3,23c)] to vary even if the materials were obtained from the same source and were crushed by the same person in the same machine. Because of the many variable factors, it can not be expected that the experience of these investigators in duplicating results would occur very often; normally, some means will be necessary for correlating results obtained with different lots if electrode carbon is used. Although any single investigator can accomplish this by running parallel tests with the same coal, no satisfactory means suggests itself for two different investigators to compare their results. For this reason, electrode carbon, as well as coke and anthracite coal, for the same objections would apply to them, do not appear to be suitable as standard inert materials.

Obviously, the ideal would be spherical grains of uniform size and of uniform surface texture. This suggests well-rounded grains of very pure silica sand. The area and character of the surface of different samples of equal weight will naturally tend to be uniform; in addition the surface

---

[2] All of the refinements added during the supplementary investigation, as far as it has gone, have been included. Kenneth A. Johnson, junior chemist, is continuing the investigation at the Northwest Experiment Station.

can be accurately measured by the methods recently proposed by Gross and Zimmerley[20] in the United States and by Martin, Bowles and Christelow[26] in England. As Kreulin[23b] and others have shown the numerical measure of the agglutinating value to depend upon the amount of surface of the inert material when the character of surface is constant, no question need arise, in the use of sand, as to whether two sets of agglutinating values obtained by the same procedure are comparable.

Against the use of sand there is the objection raised by Dunn[15] that it might fuse with constituents of the ash. This seems far-fetched in view of the low temperatures, under 1000° C., commonly employed for this test; but if it is granted that in some rare instance fusion might occur in testing a coal having an exceptionally low fusing ash, that fact can easily be detected by examining the sand after the test. The advantage of a uniform quality of surface in the inert material, and the further advantage of a surface that can be measured, will more than compensate for the trouble of examining the crushed residue from a few exceptional coals. A further objection to sand has been pointed out by Barash[3c] and Kreulin[23b], that it differs markedly from the coal in specific gravity and therefore will be more apt than a carbonaceous material to segregate in the coke buttons prior to coking. However, this is hardly a valid argument in favor of a carbonaceous material such as electrode carbon, because carbon is also appreciably higher in specific gravity than coal; hence some step for preventing segregation of the inerts between mixing and coking is needed also in procedures in which it is used. Since this step is necessary, it may as well be adapted to the use of sand.

The proper preparation of the inert material is a matter of some importance. To facilitate the preparation of duplicate samples of sand close sizing is important. This is an added advantage in the use of sand in place of carbonaceous materials. Accurate screening of friable materials such as anthracite coal, coke, or electrode carbon, especially in the smaller sizes, is impracticable. Furthermore, after friable materials are screened they are undergoing constant degradation, which is rapidly increasing the amount of surface and thus is introducing inaccuracies into the work. The exact meshes of the sieves for this purpose do not appear to be of great importance; they should be as fine as possible and as close together as practicable without rendering the screening unduly tedious and inaccurate. This condition is admirably met by the 40 and 50-mesh sieves (0.381-mm. and 0.279-mm. openings, respectively) of the W. S. Tyler Co., Cleveland, Ohio. The odd meshes are to be preferred to those in the Tyler standard screen scale, because there will be less tendency to use the sieves for other purposes than screening the test sand. Where accuracy of screening is desirable the screens should not be used for any other purpose; also, they should be frequently replaced.

A sand analyzing over 99 per cent. SiO_2 and meeting the requirement as to rounded grains (see Fig. 1) has been obtained from the Ottawa Sand Co., Ottawa, Ill. It may be purchased already screened to approximately the correct mesh, but it should be rescreened in order to have the close sizing desirable for this test. As commercial screens become worn and so deliver a product larger than the specified mesh it is desirable to order a sand having a somewhat larger range of sizes than required, as between 40 and 60 mesh.

In addition to close sizing, sand for an agglutinating test generally requires some further preparation. There is always the possibility that the surface may be coated with clay, salt, or other foreign materials.

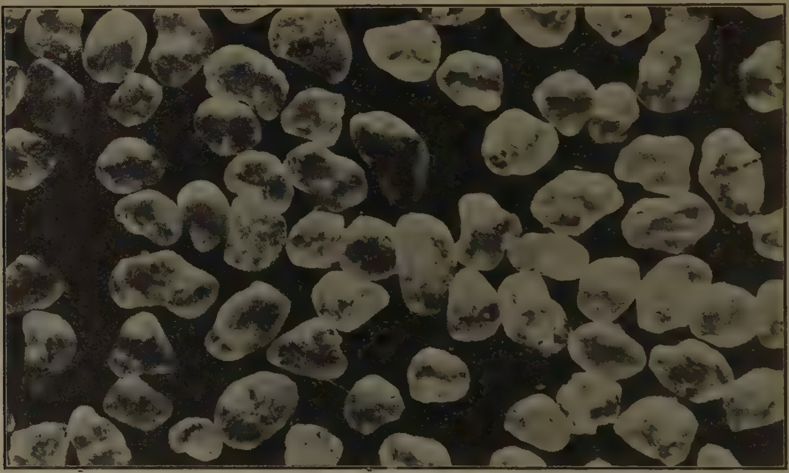


Fig. 1.—Rounded grains of sand used as inert material.

Meurice,[29] Deville[13] and others have washed their sands with dilute hydrochloric acid. Carbon tetrachloride is also satisfactory. After the sand is clean and dry it should be kept in tight cans to prevent contamination with dust and other impurities.

Preparation of Coal

The coal should be as nearly as possible in the condition in which it will ultimately be coked. If the coal is to be coked without washing it should be sampled at the tipple where it is prepared; should this be impracticable, the best procedure will generally be to secure samples from the mine. These should be taken over as large an area underground as circumstances permit, to give a correct idea of the agglutinating properties of the bed as a whole. The importance of this precaution can not be too strongly emphasized. Several instances have been found during investigations at the Northwest Station where a seam, really composed as a whole of excellent coking coal, has at certain points over

a small area given unsatisfactory test results owing to local anthracitation, or to some other cause. The sample, whether from tipple or from mine, should be large enough to be representative (see *Technical Paper* 133, U. S. Bureau of Mines, for sizes of representative samples). After it is crushed to pass a ⅛-in. screen it should be carefully mixed, several pounds should be placed in an airtight container, and the container sent promptly to the laboratory. If the coal will ultimately require washing before being used for the manufacture of coke, and it is impracticable to obtain samples of washed coal of suitable ash content, the sample must be so treated as to approximate that preparation. For this purpose the float-and-sink separation at a suitable specific gravity is convenient. As complete float-and-sink tests on a series of solutions of different specific gravities are usually available, no difficulty will ordinarily be experienced in determining the specific gravity of solution to use in floating the portion for the agglutinating test. If, for example, the float-and-sink investigations have shown that a separation at 1.50 specific gravity is required for a washed coal of suitable ash content it follows that the coal for the agglutinating test should include the portion floating on a solution of that density. As the fact is well established that a high ash content of coal decreases the strength of the coke, proper cleaning is a very necessary part of the preparation of the sample for the agglutinating test. Of course, the "float" from a float-and-sink separation is not the exact equivalent of a washed coal; but it is the nearest thing obtainable on a small scale and usually represents the washed product with sufficient accuracy.

The moisture content of the coal must be standardized if tests are to be comparative. Drying the coal at 105° C. and keeping it in tight containers prior to the test has been advocated; however, this method is faulty because coal rapidly takes up moisture from the atmosphere whenever the container is opened; hence coal prepared in this manner always has an uncertain moisture content. Moreover, the drying of fine-size coal at 105° C. is open to the objection that it increases the rapidity of oxidation, vitiating the coking properties of the coal. Nevertheless, as drying at 105° C. affords a convenient means of bringing coal to a standard moisture content, an effort has been made to overcome these objections. Oxidation has been minimized by drying the coal in the coarse condition in which it is received at the laboratory. Variations in moisture content have been prevented by allowing it, before it is ground, to come approximately to laboratory conditions, after which it is kept for 24 hr. at a temperature of about 25° C. in an atmosphere of about 40 per cent. humidity. Experience has shown that if the ground coal after such preparation is kept in a tight container, except for the exposure to the atmosphere incidental to the conduct of the agglutinating test, the moisture content will remain very uniform.

The proper grinding of the coal for this test seems to have been established in Meurice's[29a] original work. He shows agglutinating values for coals ground to pass various meshes; below 80 mesh he got nearly constant values. Besides showing that the coal should be at least as fine as that, his results also indicate that, unlike the inert material, the coal requires no further sizing. This fact is of particular importance, because a sized coal such as that used by Qvarfort,[33] Ahrens,[1] and more recently by Coffman and Layng,[11] might not contain the same proportions of the different constituents as the unsized coal. In the present test the 100-mesh Tyler standard sieve (0.147-mm. opening) has been taken as the standard opening through which the coal has been ground.

Buttons

Constancy in the weight and size of buttons for all coals is certainly a requirement of a standard test. Obviously the addition of varying amounts of inert material to a constant weight of coal could not be expected to give a uniform relationship between the agglutinating value and any property of the coke. The inequalities in the scale of agglutinating values obtained by this method have been pointed out by both Foxwell[17b] and Burdekin,[7a] as follows: ". . . the Campredon scale of caking is most unsatisfactory from a mathematical point of view. This may be readily seen by plotting the Campredon ratio against the percentage of coal in the mixture, thus:

Campredon Ratio	Coal, Per Cent.	Difference for 1 "degree"
20	4.76	0.24 per cent.
19	5.0	
5	16.6	
4	20.0	3.4 per cent.

A difference of one "degree," *i. e.* 19 to 20, or 4 to 5, represents a difference in percentage of coal from 0.24 per cent. to 3.4 per cent. in the two parts of the scale. "Thus these investigators conclude, Since there are such large differences between the values of a unit at different parts of the scale, the Campredon ratio may be very misleading."

This scale, which gives an agglutinating value relatively too high for strongly coking coals, is further distorted as a result of the larger size of buttons obtained in testing strongly coking coals. From a structural standpoint a large button containing a certain amount of agglutinant per unit weight will sustain a much heavier weight than a small button containing the same proportion of agglutinant; this fact will tend also to make the agglutinating values for strongly coking coals

relatively too high. To a lesser extent any procedure in which the proportions of inert material and coal in the buttons are varied, even though the total weight of the buttons be kept constant, is open to the objection of variations in the size of buttons, especially if the specific gravity of the inert material used in the test differs materially from that of the coal.

Variations in the proportions of inert material in the buttons are open to two further objections. The first of these is the possibility that the true maximum agglutinating value may not be found. Several investigators have reported in certain coals two maxima in the curve of crushing strength of the buttons against the percentage of inert material[13,7a] or against the amount of surface.[23b] The low agglutinating value occurring between these maxima may easily cause the first maximum to be mistaken for the maximum agglutinating value of the coal. The second of these objections is the lack of directness from an experimental standpoint of any method for determining the agglutinating value requiring a number of trials. For both these reasons as well as for the one previously discussed, an agglutinating test should be based upon constant proportions of coal and inert material.

In fixing the ratio of inert material to coal there are two considerations. The amount of inert surface should be sufficient to use all, or nearly all, of the agglutinating material in the button. But, at the same time, the button should have a measurable strength for a weakly coking coal. In fact, any test for measuring the agglutinating value should favor the weakly rather than the strongly coking coals, for its principal value lies in its usefulness in determining whether or not a coal of doubtful coking quality is suitable for some commercial project. Judged from the literature, any ratio from 10 to 17 parts of sand to one part of coal appears usable. In developing the present test, before a standard was adopted, a number of experiments were made using 5, 10, 15 and 20 parts of sand mixed with one part of coal. As 10 parts seemed to give the best proportioned scale, that ratio was adopted.

The preparation of the buttons should involve, in the first place, some step to prevent segregation of the inert materials in the interval between mixing and coking. The mere rolling of the mixture of inert material and coal while it is being poured into the container, the method advocated in most procedures, is insufficient to keep them from reseparating. When these tests were begun the materials were mixed dry, with most irregular results in the strengths of buttons. A visual inspection of the coked mixtures, even though the coal and sand had been mixed with the greatest care, showed irregular distribution of the sand. Owing to differences in specific gravity, angle of repose and coefficient of friction between sand and coal, segregation had occurred either during the pouring of the ingredients into the crucibles or while the mixtures were standing

in the crucibles subject to the slight tremors of the building. The best method of preventing segregation is to dampen the ingredients of each button with a small amount of some liquid. The amount added must be accurately measured by some means such as a standard dropper used for surface-tension experiments, because the agglutinating values have been found to decrease rapidly as the percentage of liquid is increased. Water was first used to dampen the mixture. All of the test results given later in this paper were obtained using three drops of water per button. The objection to water is the variable amount evaporating in different tests during the mixing of the buttons. To obviate this difficulty, a change was made to glycerin. At the present time one drop of glycerin weighing 0.07 g. is added to each 25-g. button.

In the second place, the preparation of the button should involve leveling the mixture of coal and sand, after they are in the crucible. This may be done with a small leveling device as shown in Fig. 2.

In the third place, some step should be provided for bringing the particles of coal and sand into intimate contact. Two methods suggest themselves, tapping the crucible and compressing the mixture. The first was rejected

Fig. 2.—Device for leveling mixture of sand and coal in crucible.

Fig. 3.—Apparatus for compressing mixture of sand and coal in crucible.

after a number of trials because it was too difficult to duplicate and because it might cause some of the sand to separate in the mixture. In working out the second the pressure has gradually been increased until the crushing strength of the buttons has approached a maximum. The pressure of 6 kilos so determined is advocated on the postulation that the agglutinating values will be more uniform and comparative than with some lower pressure. The device recommended for compressing the mixture is shown in Fig. 3.

The type of container does not seem to be a matter of great importance so long as it is a standard article of uniform dimensions and is generally available. Coors porcelain crucibles meet the requirement as to

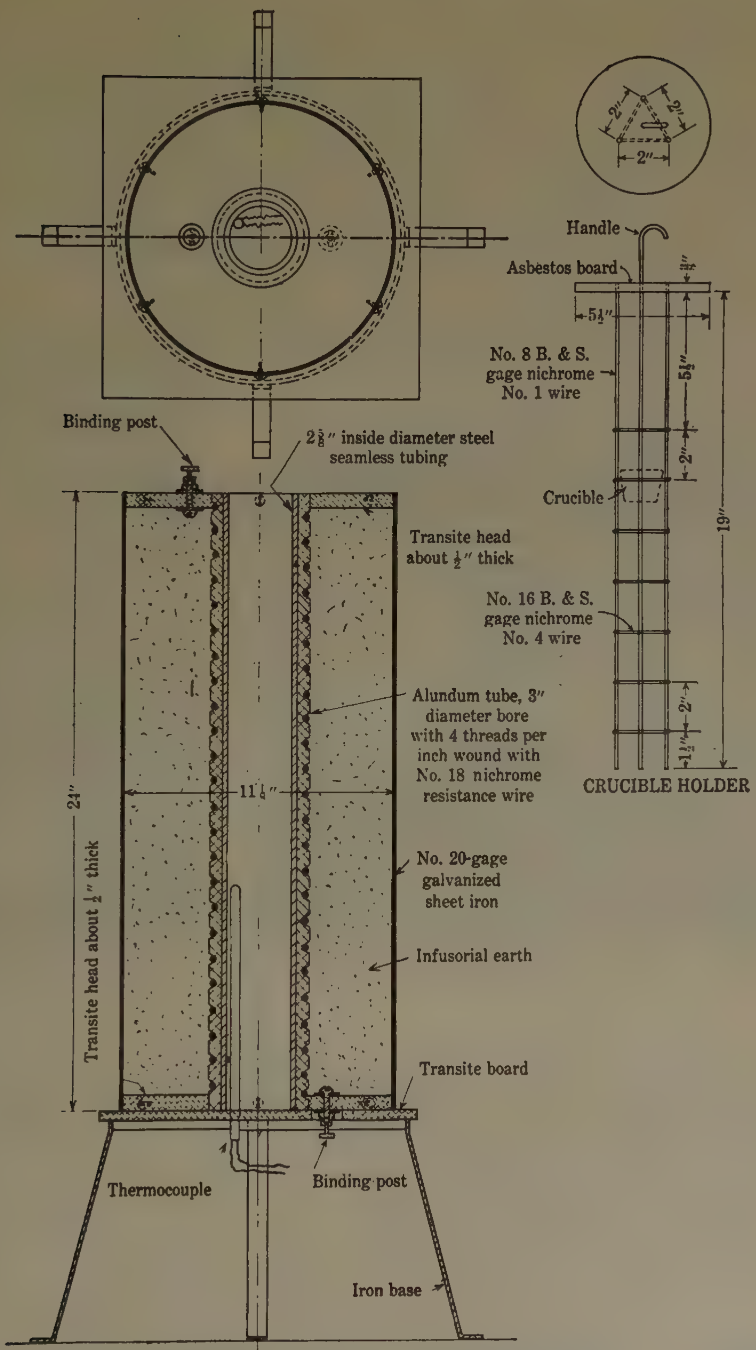


Fig. 4.—Carbonizing furnace. (Improved form developed by Pittsburgh Experiment Station of U. S. Bureau of Mines.)

availability. Uniformity of dimensions has been checked by marking a large number of crucibles so that the breaking strengths of the buttons from each could be compared. No crucible has been found with any tendency to give on the average a button of higher or lower breaking strength. The Coors high-form crucible No. 1 (height, 33.5 mm.; diameter at top, 38.0 mm.; diameter at bottom, 18.0 mm.; capacity, 25.5 c.c.) is recommended; this crucible is suitable for a 25-g. button composed of 10 parts of sand and one part of coal, which is used in the test to be described later.

For the temperature of coking, 950° C. is advocated. However, when an agglutinating test at 950° C. indicates a coke of doubtful strength, or for some other special reason, the scheme of Charpy and Godchot[9] might be tried—varying the temperature to see if a stronger coke may be made at other temperatures.

The requirements for a suitable source of heat are three: uniformity of temperature, uniform application to the crucible on all sides, and a reducing condition. The electric tube furnace with pyrometric control of the temperature certainly meets the first two requirements, and if properly constructed, the third as well. Figs. 4 and 5 show the furnace used in carbonizing the buttons. In this, five crucibles set in the wire basket are coked at one time. To insure uniform heating of the top and bottom crucibles, one blank crucible filled with sand is placed above and another below the string of five crucibles containing coal. That the heating is uniform within the requirements of the test has been verified by averaging many hundred determinations from each position in the furnace; the averages of these showed remarkable agreement with one another.

During the early part of this investigation efforts were made to get satisfactory carbonization in an electrically heated rectangular muffle furnace. Several crucibles were charged simultaneously and were carbonized at the same temperature later used with the circular furnace, but the results were irregular and could not be reproduced. A study of the irregularity showed that, owing to the differences in the rate of heat supply to the sides of the crucible, the carbonization had not proceeded uniformly, and the meeting point of the fingers which theoretically would form from the heated surface toward the center of the charge was not in the middle of the button. This was shown by the irregular failure of the buttons in the testing machines when the cones which were disclosed by the cracking of the outside cylindrical surface of the button were found to be eccentric, with the apexes farther from the point of maximum heat supply. Efforts were made to protect the outside surface of the group of crucibles by surrounding them with idle ones but this was tedious and unsatisfactory and results were still irregular. Later experience in another laboratory has shown somewhat similar results

and it is believed that a circular furnace such as is described, which will heat the crucibles uniformly around the entire circumferences, is essential for good results.

After the simultaneous charging of five buttons and two blanks, the temperature on the interior of the furnace drops about 60°. No effort is made to prevent this, the amount of current to the furnace being kept constant with an ammeter. After the first chilling of the furnace the temperature gradually returns to 950° C. and remains there during the last part of the coking period. Although the conditions in this furnace will naturally be reducing, a reducing atmosphere in the procedure to be described later is further safeguarded by the pouring on top of the mixture

FIG. 5.—CRUCIBLES BEING PLACED IN CARBONIZING FURNACE.

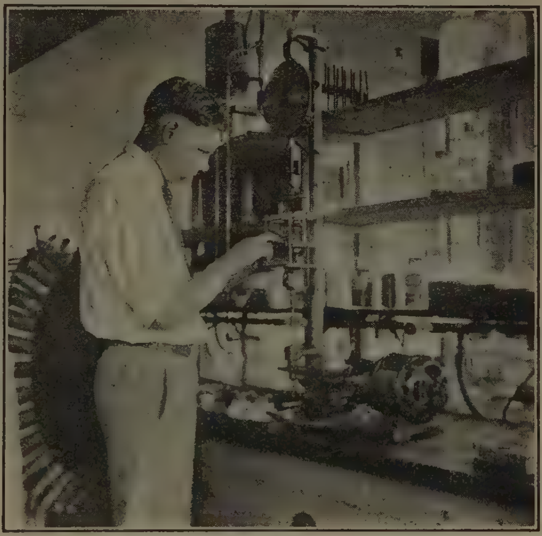


FIG. 6.—TESTING MACHINE FOR CRUSHING FINISHED BUTTONS.

in each crucible of a "blanket" of loose carbonized coal and sand from previous tests, and by placing on the crucible a porcelain cover.

The length of the period of carbonization will be fixed by the time required to bring the temperature in the furnace up to 950° C. To insure complete carbonization at that temperature, the furnace temperature must be held at 950° C. for several minutes at the end of the coking period. In coking five buttons weighing 25 g. each, a period of 20 min. has been selected as standard.

The logical test of buttons composed of a constant weight of coal and of inert material is crushing them in a testing machine. To prevent any variation in breaking strengths due to the rate of applying the breaking load the machine should be motor-driven. It should be of a type that

advances the platens at a uniform rate regardless of the resistance offered by the buttons. In most buttons two points of failure occur, a preliminary failure and an ultimate failure. The second is much the

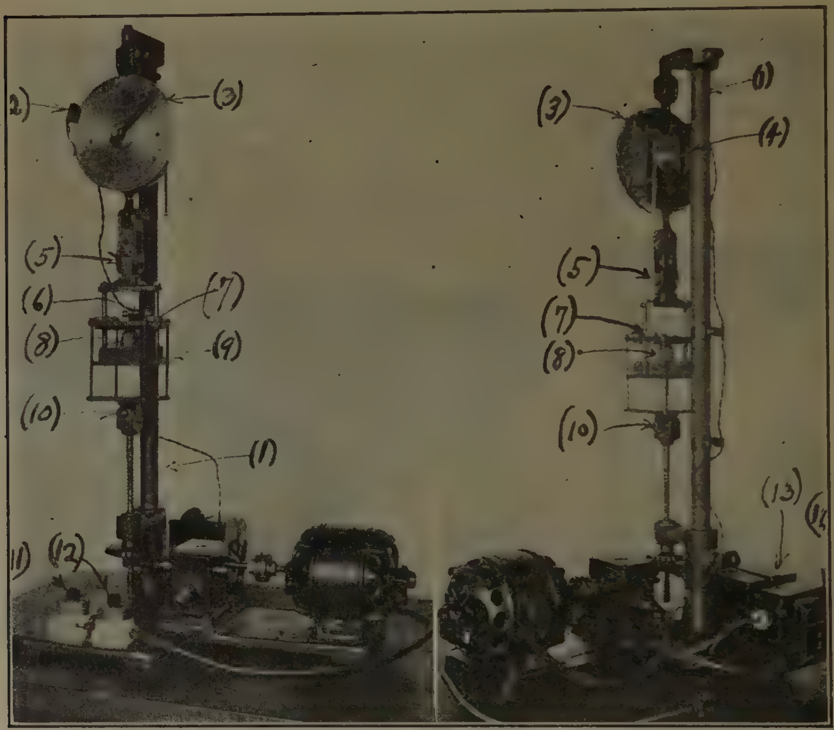


Fig. 7.—Machine for testing buttons.

1. Supporting column.
2. Movable electrical switch to stop motor if testing machine is overloaded.
3. Spring dynamometer registering to 25 kilos with all movable parts materially affecting readings equipped with roller bearings.
4. Self-locking device mounted on main central shaft of dynamometer, with lever for releasing.
5. Joint at right angles to joint at No. 10.
6. Movable electrical switch to stop motor in case top platen is raised too high.
7. Top platen, on which are mounted roller bearings to guide rods pulling lower platen. Platens touch only through these roller bearings. Rate of movement 0.048 cm. per second.
8. Button to be broken, with rubber pads or other means of equalizing pressure over top and bottom surfaces.
9. Lower platen.
10. Joint at right angles to joint at No. 5.
11. Reversing switch.
12. Stopping and starting switch.
13. Transformer supplying 11-volt current to safety switches to prevent arcing.
14. Magnetic switch to stop motor if safety switches open.

more uniform, but it is often precipitated prematurely by the first, thus introducing an irregularity into the results. This must be obviated by

selection of a type of crushing machine that will advance the platens at a uniform rate and will permit no sudden jar on the first failure, as is the case where the crushing load is applied directly by the gradual addition of weights. The design of this machine should be such that only a straight crushing force is applied; any tendency to break the buttons on an angle should be guarded against. It should automatically record the maximum crushing strength, leaving nothing to the judgment of the operator. Friction in all guides and other parts affecting the crushing strength should be minimized by the equipping of those parts with roller bearings. A machine that has proved highly satisfactory is shown in Figs. 6 and 7. This is a standard Riehle textile machine that has been modified to meet the requirements just discussed.

In breaking a button some means should be provided for equalizing the pressure over any inequalities in the surfaces in contact with the platens, especially over the slight concavity that occurs on the large end of the buttons from some coals. For this purpose soft rubber, such as is used for backing stamp pads, and thick rubber bath sponges have been tried. These have not proved entirely satisfactory, because they are sometimes softer in one spot than another and thus cause the button to break on a slight angle. As a pure hydrostatic type of pressure would evidently be the ideal, experiments are now being conducted with an equalizing device filled with liquid, the top of which is made of rubber covered with cloth.

One other point regarding the testing of the buttons should be mentioned. For some reason not understood, more regular results are obtained by allowing the buttons to stand for 24 hr. between carbonization and crushing.

Agglutinating Value

Since the button formed in a crucible has a varying cross-section, the load required to crush it cannot be calculated to a unit area, and the total force must be taken as the agglutinating value. The breaking value should be the average of at least 10 buttons, in which the allowable deviation of any individual button from the average of the 10 is small. At the present development of this procedure, a test is remade if any button differs from the average of the 10 buttons by more than 10 per cent.

Proposed Agglutinating Test

The apparatus needed is mentioned in the order of its use:

Sieves: 100-mesh Tyler standard (0.147 mm.) and 40 (0.381 mm.) and 50-mesh (0.279 mm.). The W. S. Tyler Co., Cleveland, Ohio.

Mixing crucibles: Coors high-form porcelain, glazed inside and out, size 2, capacity 57 cubic centimeters.

Carbonizing crucibles: Coors high-form porcelain, glazed inside and out, size 1, capacity 25.5 c.c., top diameter 38 mm., bottom diameter 18 mm., height 33.5 mm. (measurements on inside).

Standard surface-tension dropper (1 drop, 0.07 g.).

Special funnel with small end that will just slip inside of carbonizing crucible, used to prevent spillage.

Leveling device shown in Fig. 2.

Compression apparatus shown in Fig. 3.

Electric carbonizing furnace shown in Figs. 4 and 5.

Testing machine, Figs. 6 and 7.

Inert Material

A very satisfactory sand for use as an inert material has been obtained from the Ottawa Sand Co., Ottawa, Ill. This is a very pure silica sand composed of well-rounded grains. It is purchased already screened to approximately the correct mesh; but for greater uniformity it is carefully rescreened through 40 and on 50 mesh. After the screening it is washed with carbon tetrachloride and thoroughly dried. Surface measurements for standardizing the sand are made by the methods proposed by Gross and Zimmerley.[20] The sand at present in use has 89 sq. cm. per gram.

Preparation of Coal

A representative sample of the coal to be tested is brought to the laboratory in a sealed container. If the coal is of higher ash content than will ultimately be coked the portion to be tested is separated from the remainder with a solution of zinc chloride of the proper density to give the required ash or sulfur content. The coal for the agglutinating test, whether the "float" from a separation made with zinc chloride solution or a fresh sample from mine or washery, is dried at 105° C. and is allowed to stand 24 hr. at a temperature of 70° F. (21° C.) in an atmosphere of 40 per cent. humidity. After this preparation, which reduces the coal to a very stable moisture content, the sample is ground to pass a 100-mesh sieve and is placed in a tight container. At the time of grinding, separate samples are prepared for proximate and ultimate analyses, and for tests to determine coke and by-product yields in case those are desired.

Preparation and Testing of Buttons

Before the actual preparation of the buttons is begun, the electric furnace is turned on, so that it will be up to a temperature of 950° C. by the time the coal-sand mixtures are ready for coking. Also the surface-tension dropper is adjusted to form one drop (0.07 g.) in approximately five minutes.

After the test sand has been remixed by rolling the container, 10 lots are weighed out, each containing 22.725 g. Next, 2.275 g. of coal, likewise carefully mixed, is weighed out ready to add to the sand. One drop of glycerin from the dropper is now mixed through the sand alone and the mixing is continued for 30 sec. To this damp sand is added the weighed portion of coal, after which the whole is stirred for 60 sec. more. An efficient method of mixing the sand and glycerin, also the sand, glycerin and coal, is the following: The crucible, resting on a table, is turned with the left hand a little at a time in a counter-clockwise direction. At the same time a small spatula held in the right hand in nearly a vertical position is drawn repeatedly from the bottom of the crucible towards the top and simultaneously is moved horizontally in a clockwise direction around the side of the crucible. This method of mixing, which is very effective and rapid, will give a uniform consistency during the two prescribed intervals.

After the mixing is completed the leveling device serves to give an even surface to the contents of the crucible. Then for a period of 30 sec. the mixture is subjected to a load of 6 kilos by use of the compression apparatus, Fig. 3. The empty space left at the top of the crucible is filled with a loose blanket of carbonized coal and sand remaining from preceding tests, and, as an added precaution, a porcelain cover is laid over the top. After all 10 buttons are prepared in this manner, five at a time are placed in a rack like that shown in Figs. 4 and 5.

The buttons are carbonized, five at a time, for 20 min. at a maximum temperature of 950° C. Then they are set aside for 24 hr. before being crushed.

When the buttons are to be broken the blanket is poured off and each button is placed in an inverted position on a soft rubber pad, or other device for distributing the pressure uniformly. The top is likewise protected by a small rubber pad. The button, with its accompanying pads, is placed in the center of the platen of the testing machine and is crushed. In addition to a record of the maximum crushing strength, notes are made of any unusual feature of the test, such, for example, as the failure of a button on one side, or any other irregularity that might indicate some lack of care in the preparation. If any button differs from the average of the 10 buttons by more than 10 per cent. the test is repeated.

Agglutinating Value

The agglutinating value is the average crushing strength in grams of 10 buttons, determined in the manner just described.

In order that the reader may have an idea of the variation between individual buttons and between duplicate tests, a few typical, average tests are given in Table 2 for the standard sample of Wilkeson coal in

use in the laboratory. Tests 85 and 86 each have one button differing from the average by over 10 per cent. and so the tests would be repeated in ordinary testing. The average of tests 87, 88, 94, 95 and 98 is 8013. The maximum difference, shown in test 98, is 4.5 per cent.

On a basis of the procedure as described, any coal with an agglutinating value over 3500 will make a satisfactory coke for blast-furnace use. A very strongly coking coal such as Pocahontas No. 3 should give a value of about 8000.

Table 2.—*Typical Data Obtained in Testing a Standard Sample of Wilkeson Coal*

Test No.	85	86	87	88	94	95	98
	8,250	8,075	8,200	8,200	8,375		7,450
	8,100	8,300	8,500	8,075	8,125	7,875	7,775
	7,750	8,000	7,675	7,850	7,975	7,400	7,400
	8,700	7,900	8,475	7,875	7,750	8,475	7,900
	9,150	9,000	8,650	8,300	7,700	7,725	8,025
	7,900	7,750	8,000	7,600	8,650	8,800	8,000
	8,150	7,975	8,275	8,300	7,475	9,100	8,100
	8,200	7,500	7,700	7,400	7,900	8,350	7,800
	7,875	7,700	7,900	7,350	8,150	7,550	7,350
	10,200	8,125	8,650	7,550	8,300	8,700	7,700
Average.......	8,428	8,033	8,203	7,850	8,040	8,220	7,750

Correlations

Before agglutinating values obtained by purely empirical methods can serve any useful purpose it must be shown that they bear a definite relationship to the strength and the resistance to shattering of oven cokes produced from the same coals. In addition it is desirable, as further evidence of the dependability of the test, to develop correlations between agglutinating values and constituents of the coal that are believed to indicate their coking properties.

The data available on the test described are meager and more work is needed to establish them more fully. Furthermore, the correlations were all made before the test was in its present highly developed state, hence the relationships shown later would undoubtedly be more uniform if the tests were repeated by present methods. For instance, the buttons were formerly crushed in a homemade machine not capable of the accuracy of the one described in the procedure. Again, water with the accompanying difficulties of irregular evaporation was used to dampen the ingredients of the buttons. Before these and some less important sources of error were corrected a determination was accepted if not above three buttons

differed by over 10 per cent. from the average of the 10; whereas at present a test is remade if any button differs from the average by over 10 per cent. However, the available data indicate the possibilities of this method of determining agglutinating values and in view of the general interest at this time in a laboratory method of predicting coking strengths of coals, it is felt that the procedure should be published in its present form together with the accumulated data.

Correlations of Agglutinating Values with Physical Properties of Cokes

Before discussing these correlations it is necessary to distinguish between strength and resistance to shattering. Strength of coke, as the term is used in this paper, refers to its capacity to sustain a load, either static or slowly moving, such as it must support during descent in a blast furnace. Strength of a metallurgical coke can only be determined with certainty by its use under blast-furnace operating conditions. However, to some extent the size of pieces of coke discharged from the ovens, the coherence of individual pieces, the presence or absence of fine checks, the resistance of the pieces of coke to handling, and like properties are indications of its strength. But density, so commonly regarded as a criterion of strength of coke, is not dependable, for a coke with a high percentage of pore space may have very strong cell walls and be able to support a heavy burden. As distinguished from strength, resistance to shattering is the capacity of the coke to withstand sudden blows. A weak coke may show a high resistance to shatter due to sponginess and elasticity of the pieces; a strong coke may break readily in small pieces and thus give a poor shatter test, due to brittleness.

The correlation of the agglutinating values of coals with the capacities of cokes to withstand shattering may be made by use of the shatter or the rumbler tests. But no similar test is available for use in correlating agglutinating values with strengths of cokes. Crushing tests of cubes of coke were tried in connection with this investigation but the results, as have been found by other investigators, were too erratic to be dependable. This leaves as the only criteria of coking strength the reports of users of the cokes. Although these naturally vary, even though only the judgments of experienced users of coke be considered, they are more nearly in accord than might be supposed, and they serve as a basis for grouping cokes as strong, medium strong, or weak. Such groupings have been made of the reports of experienced users of the cokes and with them have been correlated the agglutinating values obtained by the methods described in this paper.

Two series of coals have been obtained at different times from oven plants that are manufacturing coke for use in the iron blast furnace.

TABLE 3.—*Agglutinating Values, Proximate and Ultimate Analyses and Distillation Tests of Freshly Mined Samples*

Sample[a]		Proximate Analysis			Ultimate Analysis						
	1	2	3	4	5	6	7	8	9	10	11
	Agglutinating Value, Grams	Ash, Per Cent.	Volatile, Per Cent.	Fixed Carbon, Per Cent.	S, Per Cent.	C, Per Cent.	H, Per Cent.	N, Per Cent.	O, Per Cent.	O Ash-free Basis, Per Cent.	$\frac{H}{O} = \frac{\text{Col. 7}}{\text{Col. 9}}$
Christian, N. M.	0	4.60	49.00	46.40	0.69	72.12	5.75	1.60	15.24	16.0	0.38
Roslyn 8, Wash.	160	16.10	39.10	44.80	0.42	66.46	5.09	1.23	10.70	12.8	0.48
Nanaimo, B. C.	645	9.90	40.50	49.60	0.92	71.74	5.21	1.24	10.99	12.2	0.47
Ft. Lewis, Colo.	740	10.20	37.30	52.50							
Coal H	1,210	8.30	40.20	51.50	0.66	73.49	5.32	1.53	10.70	11.7	0.50
San Juan Basin, N. M.	2,668	7.60	34.60	57.80	0.58	77.20	5.37	1.40	7.85	8.5	0.68
Coal G	3,119	10.10	34.60	55.30	1.41	75.38	5.20	1.25	6.66	7.4	0.78
Oven Test, Wash.[c]	4,299	12.20	22.96	64.84	0.54	74.15	4.89	1.90	6.32	7.2	0.77
Composite, Wash.[c]	4,303	12.60	24.00	63.40	0.52	74.26	4.57	2.30	5.75	6.6	0.80
Coal F	4,503	9.80	30.80	59.40	1.03	76.60	4.84	1.29	6.44	7.1	0.75
Coal E	4.930	8.50	25.30	66.20	1.40	79.46	4.65	1.21	4.78	5.2	0.97
Coal D	5,012	9.10	34.20	56.70	1.17	76.56	5.15	1.53	6.49	7.1	0.79
Coal C	5,430	10.80	13.20	76.00	1.26	76.42	4.75	1.32	5.45	6.1	0.87
Wilkeson 8, Wash.	5,563	9.98	28.82	61.20							
Coal B	6,011	6.10	26.90	67.00	1.05	81.84	4.86	1.49	4.66	5.0	1.04
Coal A	6,570	3.20	18.50	78.30	0.59	87.43	4.58	1.24	2.96	3.1	1.55
Fairfax 2, Wash.	6,795	9.92	22.42	67.66							
Wilkeson 7, Wash.	7,001	12.88	26.78	60.34							

[a] Sources of coals: A, high-grade Pocahontas, West Virginia; B, Pratt seam, Alabama; C, a mixture of coals from Upper Kittanning, Freeport, and Pittsburgh seams, Pennsylvania, and Pocahontas, West Virginia; D, Pittsburgh seam, Pennsylvania; E, a mixture of coals from Upper and Lower Kittanning and Pittsburgh seams, Pennsylvania; F, a mixture of coals from Pittsburgh seam from Greene County, Pennsylvania, and Pocahontas No. 3 from McDowell County, W. Va.; G, a mixture of coals from three seams mined in Las Animas County, Colo.; H, coal from the Mesa Verde formation at Sunnyside, Utah.

[b] Of coal.

[c] Average samples of large undeveloped area in Pierce County, including some noncoking seams.

TABLE 3.—(*Continued*)

Sample[a]	Ultimate Analysis		Distillation Results					Gas Analysis			
	12	13	14	15	16	17	18	19	20	21	22
	$\frac{C}{O} = \frac{\text{Col. 6}}{\text{Col. 9}}$	$H - \frac{O}{8}$ on Ash-free Basis, Per Cent.	Coke Yield, Per Cent.	Sulfate, per Ton, Lb.	Light Oil per Ton, Gal.	Tar per Ton, Gal.	Gas (Wet, 62° F., 760 mm.) per Ton, Cu. Ft.	CO_2, Vol., Per Cent.	Illuminants, Vol., Per Cent.	CO, Vol., Per Cent.	H_2, Vol., Per Cent.
Christian, N. M.	4.73	3.75	62.70	37.13	4.68	5.72	13,039	6.32	3.89	14.01	48.00
Roslyn 8, Wash.	6.21	3.49	66.33	34.76	3.00	11.13	10,440	5.40	4.83	13.30	40.40
Nanaimo, B. C.	6.53	3.68	66.92	31.37	4.77	7.01	11,251	5.62	3.95	13.83	46.75
Ft. Lewis, Colo.			72.24	26.85	4.09	1.61	12,604	4.09	1.24	11.76	52.27
Coal H.	6.87	3.86	70.96	28.16	4.33	3.05	12,258	3.91	1.73	12.00	50.45
San Juan Basin, N. M.	9.83	4.31	74.76	23.78	4.04	3.19	11,642	2.42	2.18	8.77	44.20
Coal G.	11.32	4.28	74.80	24.77	3.92	4.08	11,270	2.71	2.22	8.23	53.39
Oven Test, Wash.[c]	11.74	3.99	80.16	30.37	2.19	1.93	11,260	2.68	1.39	6.66	55.17
Composite, Wash.[c]	12.92	3.75	79.25	33.90	1.80	4.20	10,415	3.19	2.03	7.06	56.45
Coal F.	11.90	3.95	76.74	22.29	3.53	4.71	11,279	2.39	1.83	7.99	57.37
Coal E.	16.62	4.00	80.42	21.47	2.69	3.82	10,973	2.26	1.63	5.59	62.26
Coal D.	11.80	4.26	76.88	24.34	4.03	3.65	12,000	2.40	1.30	7.50	58.00
Coal C.	14.03	3.99	79.06	22.72	3.23	4.83	10,843	1.73	1.78	6.42	57.40
Wilkeson 8, Wash.			77.59	33.72	3.13	5.14	11,560	1.83	2.16	6.28	58.51
Coal B.	17.57	4.23	80.00	23.81	3.11	4.54	11,504	2.20	1.50	4.70	63.00
Coal A.	29.55	4.19	84.96	18.90	1.87	2.10	11,007	1.08	1.10	4.45	63.89
Fairfax 2, Wash.			82.74	28.23	2.33	2.31	10,952	1.29	1.09	4.54	61.79
Wilkeson 7, Wash.			79.03	34.10	2.86	3.37	11,330	1.87	1.67	5.49	57.21

[a] Sources of coals: A, high-grade Pocahontas, West Virginia; B, Pratt seam, Alabama; C, a mixture of coals from Upper Kittanning, Freeport, and Pittsburgh seams, Pennsylvania, and Pocahontas, West Virginia; D, Pittsburgh seam, Pennsylvania; E, a mixture of coals from Upper and Lower Kittanning and Pittsburgh seams, Pennsylvania; F, a mixture of coals from Pittsburgh seam from Greene County, Pennsylvania, and Pocahontas No. 3 from McDowell County, W. Va.; G, a mixture of coals from three seams mined in Las Animas County, Colo.; H, coal from the Mesa Verde formation at Sunnyside, Utah.

[b] Of coal.

[c] Average samples of large undeveloped area in Pierce County, including some noncoking seams.

TABLE 3.—(*Continued*)

Sample[a]	Gas Analysis										
	23	24	25	26	27	28	29	30	31	32	33
	CH_4, Vol., Per Cent.	N, Vol., Per Cent.	CO_2, Weight, Per Cent.[b]	CO, Weight, Per Cent.[b]	O in CO_2, Weight, Per Cent.[b]	O in CO, Weight, Per Cent.[b]	O in CO_2 + CO, Weight, Per Cent.[b]	O in H_2O, Weight, Per Cent.[b]	O in CO_2 + CO + H_2O Weight, Per Cent.[b]	O in CO_2 + CO + H_2O Ash-free Basis, Per Cent.[b]	O in CO + CO_2 Ash-free Basis, Per Cent.[b]
Christian, N. M.	25.89	1.89	4.65	6.50	3.38	3.71	7.09	8.11	15.20	15.9	7.4
Roslyn 8, Wash.	30.12	5.95	3.19	4.94	2.32	2.82	5.14	7.43	12.57	15.0	6.1
Nanaimo, B. C.	29.29	0.56	3.60	5.58	2.62	3.19	5.81	6.59	12.40	13.8	6.4
Ft. Lewis, Colo.	26.71	3.93	3.00	5.20	2.18	2.97	5.15	5.96	11.11	12.4	5.7
Coal H.	27.58	4.33	2.80	5.26	2.04	3.01	5.05	6.39	11.44	12.5	5.5
San Juan Basin, N. M.	36.05	6.38	1.51	3.63	1.10	2.07	3.17	4.86	8.03	8.7	3.4
C al G.	29.88	3.57	1.76	3.31	1.28	1.89	3.17	4.89	8.06	9.0	3.5
Oven Test, Wash[c]	26.25	7.90	1.71	2.66	1.24	1.52	2.76	3.93	6.69	7.6	3.1
Composite, Wash.[c]	25.03	6.24	1.89	2.62	1.37	1.50	2.87	4.05	6.92	7.9	3.3
Coal F.	25.80	4.62	1.55	3.21	1.13	1.83	2.96	4.70	7.66	8.5	3.3
Coal E.	23.33	4.93	1.42	2.17	1.03	1.24	2.27	3.60	5.87	6.4	2.5
Coal D.	25.20	5.60	1.62	3.20	1.18	1.83	3.01	4.09	7.10	7.8	3.3
Coal C.	24.91	7.76	1.08	2.47	0.79	1.41	2.20	3.89	6.09	6.8	2.5
Wilkeson 8, Wash.	29.25	1.97	1.23	2.59	0.89	1.48	2.37	3.86	6.23	6.9	2.6
Coal B.	23.10	5.50	1.43	1.93	1.04	1.10	2.14	2.94	5.08	5.4	2.3
Coal A.	25.84	3.64	0.70	1 75	0.51	1.00	1.51	2.20	3.71	3.8	1.6
Fairfax 2. Wash.	26.26	5.03	0.83	1.78	0.60	1.02	1.62	2.99	4.61	5.1	1.8
Wilkeson 7, Wash.	31.11	2.65	1.24	2.20	0.90	1.26	2.16	3.78	5.94	6.8	2.5

[a] Sources of coals: A, high-grade Pocahontas, West Virginia; B, Pratt seam, Alabama; C, a mixture of coals from Upper Kittanning, Freeport, and Pittsburgh seams, Pennsylvania, and Pocahontas, West Virginia; D, Pittsburgh seam, Pennsylvania; E, a mixture of coals from Upper and Lower Kitanning and Pittsburgh seams, Pennsylvania; F, a mixture of coals from Pittsburgh seam from Greene County, Pennsylvania, and Pocahontas No. 3 from McDowell County, W. Va.; G, a mixture of coals from three seams mined in Las Animas County, Colo.; H, coal from the Mesa Verde formation at Sunnyside, Utah.

[b] Of coal.

[c] Average samples of large undeveloped area in Pierce County, including some noncoking seams.

TABLE 4.—*Agglutinating Values, Proximate and Ultimate Analyses and Distillation Tests of Freshly Mined Samples*

Sample[a]		Proximate Analysis			Ultimate Analysis						
	1	2	3	4	5	6	7	8	9	10	11
	Agglutinating Value, Grams	Ash, Per Cent.	Volatile, Per Cent.	Fixed Carbon, Per Cent.	S, Per Cent.	C, Per Cent.	H, Per Cent.	N, Per Cent.	O, Per Cent.	O, Ash-free Basis, Per Cent.	$\frac{H}{O} = \frac{\text{Col. 7}}{\text{Col. 9}}$
Coal 1	2,301	6.68	41.01	52.31	1.09	74.68	5.23	1.58	10.74	11.51	0.49
Coal 2	3,908	3.41	40.92	55.67	0.99	79.77	5.91	1.46	8.46	8.76	0.70
Coal 3	4,870	1.33	38.35	60.32	0.59	83.12	5.50	1.63	7.83	7.94	0.70
Wilkeson No. 2, Wash	6,773	12.89	29.43	57.68	0.69	74.56	5.19	2.38	4.29	4.92	1.21
Coal 4	7,098	1.75	26.60	71.65	0.52	87.43	5.24	1.94	3.12	3.18	1.68
Wilkeson No. 1, Wash	7,603	7.91	32.70	59.39	0.41	77.80	5.14	2.58	6.16	6.69	0.83
Coal 5	7,826	3.35	35.51	61.14	0.60	82.45	5.49	1.34	6.77	7.00	0.81
Coal 6	7,838	4.83	18.02	77.15	0.52	85.87	4.25	1.13	3.40	3.57	1.25
Coal 7	8,557	1.05	20.60	78.35	0.52	89.05	4.65	1.63	3.10	3.13	1.50
Fairfax, Wash	9,109	14.15	21.92	63.93	0.46	75.30	4.74	1.88	3.47	4.04	1.37
Coal 8	9,253	10.08	31.49	58.43	0.79	78.38	4.68	1.06	5.01	5.57	0.93

[a] Sources of Coals: Coal 1 and Coal 2, Mesa Verde formation, Sunnyside, Utah. Coal 3, seam name unknown. Coal 4, Sewell seam, Fayette County, W. Va. Coal 5, Taggart seam, Wise County, Va. Coal 6, Pocahontas No. 3 seam, McDowell County, W. Va. Coal 7, Fire Creek seam, Fayette County, W. Va.

[b] Of coal.

TABLE 4.—(*Continued*)

Sample[a]	Ultimate Analysis		Distillation Results					Gas Analysis			
	12	13	14	15	16	17	18	19	20	21	22
	$\frac{C}{O} = \frac{\text{Col. 6}}{\text{Col. 9}}$	$H - \frac{O}{8}$ O on Ash-free Basis, Per Cent.	Coke Yield, Per Cent.	Sulfate, per Ton, Lb.	Light Oil per Ton, Gal.	Tar per Ton, Gal.	Gas (Wet, 62° F., 760 Mm.) per Ton Cu. Ft.	CO_2, Vol., Per Cent.	Illuminants, Vol., Per Cent.	CO, Vol., Per Cent.	H_2, Vol., Per Cent.
Coal 1	6.95	3.79	67.75	27.5	3.48	7.01	12,020	3.6	3.8	11.2	45.8
Coal 2	9.43	4.81	68.09	24.1	3.41	11.98	11,130	3.5	4.1	10.4	46.7
Coal 3	10.62	4.51	69.81	28.1	3.59	10.67	11,860	2.0	3.6	8.6	50.3
Wilkeson No. 2, Wash	17.38	4.57	75.66	32.5	3.17	5.31	11,255	1.9	3.6	5.7	51.0
Coal 4	28.02	4.84	76.65	29.5	2.12	6.12	12,395	0.9	2.8	4.5	53.0
Wilkeson No. 1, Wash	12.63	4.30	73.42	34.5	2.32	4.93	11,600	2.6	3.6	6.8	49.6
Coal 5	12.18	4.61	72.65	22.6	3.99	8.14	11,925	2.1	3.7	8.3	49.1
Coal 6	25.26	3.80	85.13	18.8	1.26	3.23	10,350	0.9	1.5	3.7	65.3
Coal 7	28.73	4.26	80.77	23.8	1.77	4.49	11,955	0.7	2.3	3.9	59.5
Fairfax, Wash	21.70	4.23	82.14	25.4	1.26	7.40	9,705	1.5	2.8	4.5	59.3
Coal 8	15.64	3.98	75.55	17.6	2.97	6.94	10,870	2.9	3.5	6.8	51.3

[a] Sources of Coals: Coal 1 and Coal 2, Mesa Verde formation, Sunnyside, Utah. Coal 3, seam name unknown. Coal 4, Sewell seam, Fayette County, W. Va. Coal 5, Taggart seam, Wise County, Va. Coal 6, Pocahontas No. 3 seam, McDowell County, W. Va. Coal 7, Fire Creek seam, Fayette County, W. Va.

[b] Of coal.

TABLE 4.—(*Continued*)

Sample[a]	Gas Analysis										
	23	24	25	26	27	28	29	30	31	32	33
	CH_4, Vol., Per Cent.	N, Vol., Per Cent.	CO_2, Weight, Per Cent.[b]	CO, Weight, Per Cent.[b]	O in CO_2, Weight, Per Cent.[b]	O in CO, Weight, Per Cent.[b]	O in CO_2 + CO, Weight, Per Cent.[b]	O in H_2O, Weight, Per Cent.[b]	O in CO_2 + CO + H_2O Weight, Per Cent.[b]	O in CO_2 + CO + H_2O Ash-free Basis, Per Cent.[b]	CO + CO_2 Ash-free Basis, Per Cent.[b]
Coal 1	30.2	5.4	2.46	4.87	1.79	2.78	4.57	6.45	11.02	11.81	4.90
Coal 2	30.4	4.9	2.21	4.18	1.61	2.39	4.00	4.70	8.70	9.01	4.14
Coal 3	31.7	3.8	1.35	3.69	0.98	2.11	3.09	5.66	8.75	8.87	3.13
Wilkeson No. 2, Wash	28.7	9.1	1.22	2.32	0.89	1.33	2.22	3.40	5.62	6.45	2.55
Coal 4	31.9	6.9	0.63	2.02	0.46	1.15	1.61	2.47	4.08	4.15	1.64
Wilkeson No. 1, Wash	28.4	9.0	1.71	2.85	1.24	1.63	2.87	4.74	7.61	8.26	3.12
Coal 5	31.2	5.6	1.42	3.58	1.03	2.05	3.08	4.13	7.21	7.46	3.19
Coal 6	24.6	4.0	0.53	1.38	0.39	0.79	1.18	2.10	3.28	3.45	1.24
Coal 7	27.1	6.5	0.48	1.69	0.35	0.97	1.32	1.97	3.29	3.32	1.33
Fairfax, Wash	26.5	5.4	0.83	1.58	0.60	0.90	1.50	1.61	3.11	3.62	1.75
Coal 8	26.6	8.9	1.79	2.67	1.30	1.53	2.83	3.18	6.01	6.68	3.15

[a] Sources of Coals: Coal 1 and Coal 2, Mesa Verde formation, Sunnyside, Utah. Coal 3, seam name unknown. Coal 4, Sewell seam, Fayette County, W. Va. Coal 5, Taggart seam, Wise County, Va. Coal 6, Pocahontas No. 3 seam, McDowell County, W. Va. Coal 7, Fire Creek seam, Fayette County, W. Va.

[b] Of coal.

In the selection of samples an effort was made to secure coals having a range of coking strengths. Furthermore, as strength of coke is so largely a matter of personal judgment, preference was given to coals or mixtures of coals having well-known coking properties. The results of testing these coals are recorded in Tables 3 and 4. The coals designated by the letters A, B, C, etc., in Table 3 and by numerals in Table 4 are those yielding cokes of known properties; the others are representative samples of freshly mined, Western coals. The data upon which the correlations are based include a summary of the available information as to the strength of the cokes, shatter tests of certain of the cokes, proximate and ultimate analyses of the coals and distillation tests by the methods of the United States Steel Corpn.[27] Because of a change in the procedure during the interval of several years elapsing between the testing of the two series of type coals, the agglutinating values in Table 4 must be multiplied by a factor of 0.7 to render them comparable to those in Table 3.

Agglutinating Value and Strength of Coke

The information available on the coking strength of the coals in Tables 3 and 4 leads to the belief that the agglutinating values of the type coals quite accurately represent their relative coking strengths. That is, if the coals were arranged by a body of experienced users of coke in the order of the strengths of cokes manufactured from them, they would be in the same order as given by the agglutinating values. For example, coal A, Table 3, from the Pocahontas district, West Virginia, makes one of the strongest cokes used in this country. Coal B, next in point of agglutinating value, is a standard coking coal from Alabama, although it is not considered to make a coke quite as strong as that from coal A. At the other end of the scale coal H, a Mid-Western coal, yields probably the weakest coke used in American blast-furnace practice. Coal G, from the same section of the country, gives also a comparatively weak coke, though it is distinctly stronger than that from coal H. Coals C, D, E and F, with agglutinating values varying between 4503 and 5430, are considered to yield excellent blast-furnace cokes of medium strength. A like study of the coking properties of the seven type coals in Table 4 indicates that the agglutinating test, if it does not actually give correctly the relative coking strengths of cokes manufactured from these coals, at least supplies a basis for the correct classification of the coals into general groups as strong, medium, or weak coking coals. Furthermore, the regular decrease of agglutinating values to a low limit, which corresponds to the weakest coke usable in the blast furnace, indicates that the test supplies a means of predicting whether or not by present methods of coke manufacture a coke can be made of sufficient strength for use in the iron blast furnace.

This uniform increase of coking strengths with increase of agglutinating values is not in accord with the findings of some earlier investigators who report that certain coals having very high percentages of agglutinants did not make strong cokes.[3,18] Instances are cited in which the addition of some poorly coking coal to a coal having an excess of agglutinant actually improved the coke strength. No instance of this kind has been found among the samples thus far tested. Without exception, a very high agglutinating value has been associated with a very strong coke.

In passing, it is interesting to note the agglutinating values of the coking coals available in Pierce County, Wash. The agglutinating values for Wilkeson 7 and Fairfax 2 indicate a higher coking strength than that of coal A, representing Pocahontas coal, and Wilkeson 8 is only slightly below the Pocahontas.

Agglutinating Value and Resistance to Shattering

Unfortunately, results of shatter tests are available for only four of the cokes manufactured from these coals and for one other coal that was omitted because the ultimate analysis was manifestly incorrect. The results of the tests are as follows:

Coal	Agglutinating Value	Shatter Test	
		On 2 In.	Through 2 In.
Coal H	1210	0.00	100.00
Coal G	3119	67.00	33.00
Coal E	4930	79.34	20.66
Coal C	5430	72.00	28.00
Seattle Lighting Co	5863	48.46	51.54

The curve in Fig. 8 shows the relation of the agglutinating value to the shatter test for these five coals. This indicates a very definite relation between these two tests when the coal is carbonized for the purpose of producing blast-furnace coke. At Seattle the ovens are run with the object of producing illuminating gas rather than of making strong coke, and the coke is a by-product; the shatter test cannot be expected to be comparable with that from the other plants. While Fig. 8 shows a definite relation between these two factors there are too few points to make possible definite conclusions regarding this relationship, and further experience showing the variation of the shatter test with the agglutinating values will be required before a final opinion is possible. A valuable comparison might also be made between the agglutinating values and the results of the rumbler test; but no data are available.

Perhaps some members of the Institute may have data which they can submit in the form of discussion showing the relation between oxygen

content of coals and the rumbler tests of cokes produced from them. It would be interesting to make such a comparison and then to correlate the results with the uniform relationship thus far between the oxygen content of coal and its agglutinating value as shown later. These data could supplement those given herein until more complete data can be obtained.

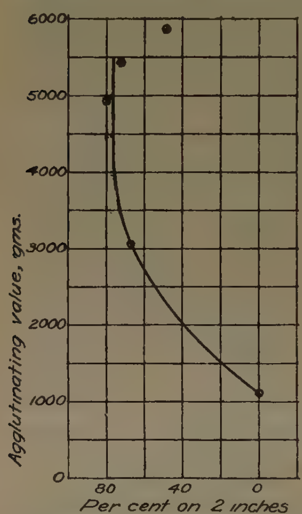


FIG. 8.—RELATIONSHIP BETWEEN AGGLUTINATING VALUE OF COAL AND SHATTER TEST OF COKE.

CORRELATIONS OF AGGLUTINATING VALUES WITH CONSTITUENTS OF COALS

Inasmuch as consistently good relationships apparently exist between the physical properties of cokes and the agglutinating values of the coals, it will be of interest to study the relationship of agglutinating values to the oxygen content of coals, to the $\frac{H}{O}$ ratio, or to the $\frac{C}{O}$ ratio, all of which have been proposed by various investigators as factors for predicting the coking properties of coals. Before these correlations are discussed, however, it will be well to point out their probable limitations in the manner proposed by Rose.[36] Using stippling to represent the limits within which most analyses fall, he has plotted the triaxial diagram of Ralston,[34] as shown in Fig. 9. To this he has added lines, as shown in Fig. 10, representing the various ratios. His

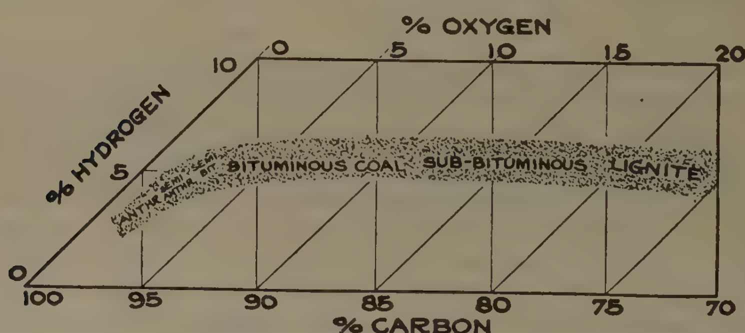


FIG. 9.—TRIAXIAL DIAGRAM OF ULTIMATE ANALYSES OF COALS (H. J. ROSE[36b]).

illuminating explanation of these diagrams is quoted below, the figure numbers having been changed to correspond to those in this paper.

" . . . owing to the sharp curvature in the band of coal analyses occurring at about 92 per cent. carbon, five per cent. hydrogen, and three

per cent. oxygen, no linear equation can be adequate over the whole range of coals.

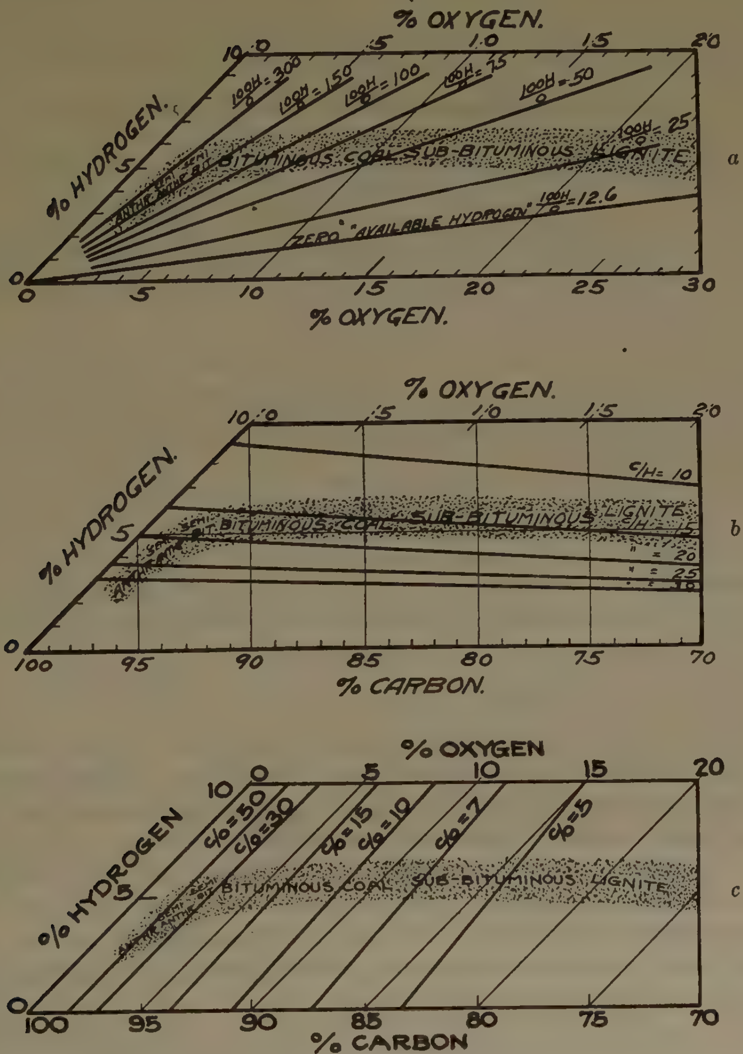


Fig. 10.—Triaxial diagrams showing relationship of various ratios to analysis of average coals (H. J. Rose[36b]).

"Thus in Fig. 10*a*, the lower $\frac{H}{O}$ ratios cut across the band of lignite, subbituminous, and bituminous coals (though much more obliquely than the oxygen lines), but for the coals of higher rank, ratio lines run *parallel* with the coal band. Thus, a single $\frac{H}{O}$ ratio line may pass through as

many as four ranks of coal (as well as cannel coals, which are not shown but would fall above the bituminous coals). The $\frac{H}{O}$ ratio is particularly well known because of White's frequently quoted conclusions[39] on the limits of this ratio for coals that can be coked in beehive ovens. White pointed out that the relation between $\frac{H}{O}$ ratio and coking quality was subject to exceptions for the coals of lower volatile matter content.

"Fig. 10*a* also permits the graphic estimation of the 'available hydrogen' in coal, that is, the percentage of hydrogen in excess of that which would be required to combine with all of the oxygen present to form water. This index has long been known but is generally considered to be of doubtful reliability.

"Fig. 10*b*' shows that the $\frac{C}{H}$ ratio lines are not very different in slope from the hydrogen lines, and apparently for most purposes the hydrogen percentage in pure coal would be just as satisfactory an index, and one simpler to use and understand. The $\frac{C}{H}$ ratio has been chiefly used in districts where anthracite, semianthracite and semibituminous coals are produced, for which coals this ratio serves as a means of differentiation just as the hydrogen percentage would. For coals of lower ranks the ratio would appear to have a very limited use, as it does not distinguish between coals of widely different rank.

"Fig. 10*c* shows that the slope of the $\frac{C}{O}$ ratio lines is so nearly the same as that of the oxygen lines that the use of the oxygen percentage in pure coal would probably be just as satisfactory for most purposes."

From the foregoing discussion one would expect the most consistent relationship between the agglutinating values of the coal and those ultimate constituents or ratios presented by lines that cut most nearly across the average coal line, especially the portion from the bend to about 15 per cent. oxygen, within which limits Ralston[34] found his coking coals to fall. In other words, one might expect a uniform relationship between the agglutinating values and the oxygen content or the $\frac{C}{O}$ ratio, and a somewhat less satisfactory relationship using the $\frac{H}{O}$ ratio.

As all of the data in Tables 3 and 4 were obtained from freshly mined samples, they have been used in the correlations about to be discussed. Those from Table 3 were plotted first and the curves drawn; then the values from Table 4, multiplied by a factor of 0.7, were added, using the x-symbol to represent the points.

In any correlation of the oxygen content of a coal with its agglutinating value, certain factors should be considered. The first is the marked

variability of the ash contents of the coals shown in Tables 3 and 4. If the percentage of oxygen "as determined" is plotted against the agglutinating value in coals of high ash content, it will be relatively too low; likewise the agglutinating values will be low. These tendencies are opposed, for a low oxygen content tends within limits to correspond to a high agglutinating value. This inconsistency, so far as the oxygen content is concerned, can be remedied by calculation of the analyses to an ash-free basis. However, because only a few correlations between ash content and agglutinating value have thus far been worked out, too few data are available for a similar correction of the agglutinating values. For this reason, the agglutinating values "as determined," must be plotted against the oxygen content on an "ash-free basis."

Another factor to be considered is the difficulty of accurately determining the oxygen content of coal. An unfortunate feature of an ultimate analysis of coal is the fact that the percentage of oxygen is found by difference and hence it contains the cumulative effect of all the errors in the analysis. Furthermore, no means is ordinarily afforded for checking this determination. In the results shown in Tables 3 and 4, however, because of the distillation test, an independent check is possible by calculation of the oxygen in the products of the distillation test. This result is also given in Tables 3 and 4, columns 31 and 32. As the two figures for the percentage of oxygen do not agree closely in a number of instances, two separate curves have been plotted, as given in Figs. 11 and 12. These show, throughout the range of oxygen contents covered by the experiments, a very uniform increase of agglutinating value with decrease in oxygen content. Had the experiments been extended to include coals of lower oxygen content, say of 1.5 per cent. and lower, the agglutinating values because of anthracitation would, of course, have dropped rapidly as indicated by the dotted lines.

Now the oxygen content used in the correlation, which has been calculated moisture-free on the basis of a moisture determination at 105° C., undoubtedly contains some oxygen from moisture not removed at that temperature. This fact has suggested, as a further study of the effect of oxygen on the agglutinating value of the coal, that another oxygen content be calculated from the oxygen in the CO plus the CO_2 in the gas, which will contain no moisture. This oxygen content, after being corrected to an ash-free basis, was plotted against the agglutinating values, but this relationship did not appear to be any more uniform than that given in Figs. 11 and 12.

The relationship between agglutinating values and oxygen contents has thus far been based upon fresh samples mined from a comparatively large area underground. The question is: Will the uniform relationship shown between oxygen content and agglutinating value hold for samples

of coking coals taken under other conditions? An answer has been obtained in the opening of a new area of coal in Pierce County, Wash., where a number of seams were sampled over small areas near the surface. When the agglutinating values of these samples are plotted against the oxygen contents, the result is the "shot-gun" arrangement of

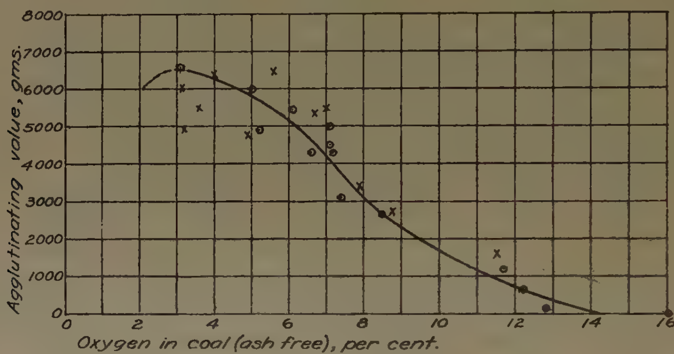


FIG. 11.—RELATIONSHIP OF AGGLUTINATING VALUE TO OXYGEN CONTENT OF COAL, WITH OXYGEN DETERMINED BY ULTIMATE ANALYSIS.

points shown in Fig. 13. As explanations of this apparent anomaly, two possibilities suggest themselves. One is that the coal was weathered in many instances, in spite of the fact that each opening in the different seams was mined until all signs of weathering disappeared. This is

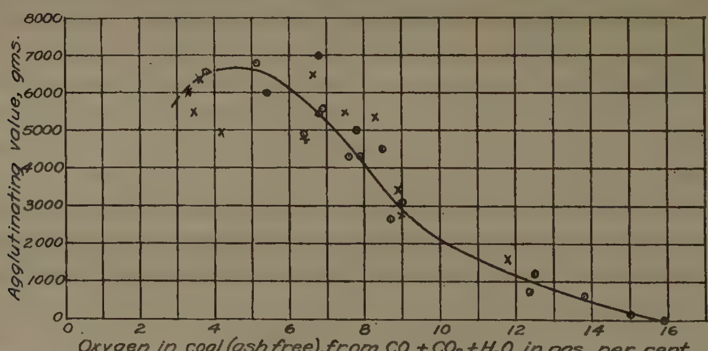


FIG. 12.—RELATIONSHIP OF AGGLUTINATING VALUE TO OXYGEN CONTENT OF COAL, WITH OXYGEN DETERMINED BY CALCULATION FROM $CO + CO_2 + H_2O$ IN GAS FROM DISTILLATION TEST.

scarcely a plausible explanation, for, in view of the relationships in Figs. 11 and 12, one would expect a fairly uniform relationship between weathered or slightly weathered coals and agglutinating values. The other is that the irregular points represent samples taken in disturbed areas. Such was found to be the case. Practically all of the points

furthest from the average curve in Fig. 13 represent samples taken near faults, on the peaks of steep anticlines, or in other similarly disturbed areas. The fact that a number of small noncoking areas have also been found in certain seams deep beneath the surface and surrounded on all sides by excellent coking coal is a further evidence that local metamorphism is the probable explanation of these irregularities. But whether the cause be metamorphism or oxidation, Fig. 13 clearly shows that certain disturbing influences affect the agglutinating value of coal without proportionately influencing its content of oxygen.

Because many of these samples were taken from seams that on the average are excellent coking coals, many of these values if taken alone would give incorrect information on the coking properties of the seam

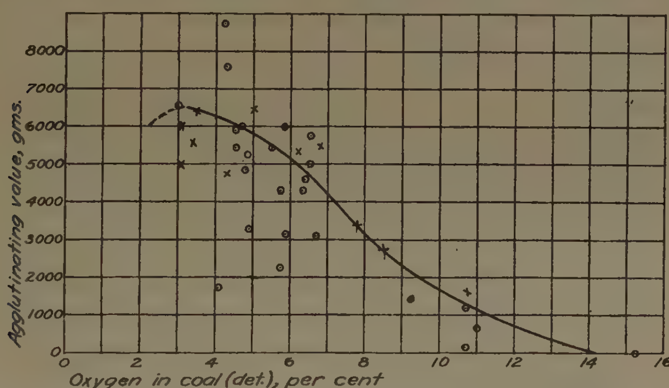


FIG. 13.—RELATIONSHIP OF AGGLUTINATING VALUE OF COAL TO OXYGEN CONTENT, SHOWING MARKED VARIATIONS INTRODUCED BY INCLUSION OF SAMPLES FROM SMALL AREAS.

as a whole. For this reason the recommendation has been made, earlier in the paper, that samples for agglutinating tests be taken over a large area underground.

Inasmuch as agglutinating values of freshly mined samples in Tables 3 and 4 show a uniform relationship with oxygen content, one would likewise expect from Rose's diagram, Fig. 10*c*, a similarly uniform relationship with $\frac{C}{O}$. Fig. 14 conforms very well to this expectation.

This study was confined almost entirely to coking coals or to coals that might be expected to coke, thus nothing in the foregoing discussion indicates that oxygen content by itself is a criterion of coking strength, but rather that, if a coal possesses coking properties, its oxygen content gives at least a good indication of the probable strength of the cokes manufacturable from it. A few of the coals tested showed no coking tendency and their oxygen contents were not determined by ultimate analysis. When they were distilled, however, the CO contents of the gas,

which will be discussed later in relation to agglutinating value, indicated a high oxygen content in the coal. Thus the facts developed all conform with the generally recognized facts that high oxygen content in coal is associated with weak coke.

The other relationship shown by Rose's diagrams as having the best possibility for predicting the coking strength of coals is the $\frac{H}{O}$ ratio. In discussing this ratio many years ago, White[39] said, "The adaptability of a coal to coking by the ordinary process appears to be indicated with a fair degree of certainty by the ratio of the hydrogen to the oxygen, moisture-free basis. Practically all coals with $\frac{H}{O}$ ratios of 59 (per cent.) or over seem to possess the quality of fusion and swelling necessary to good coking. Most coals with ratios down to 55 will make coke of some

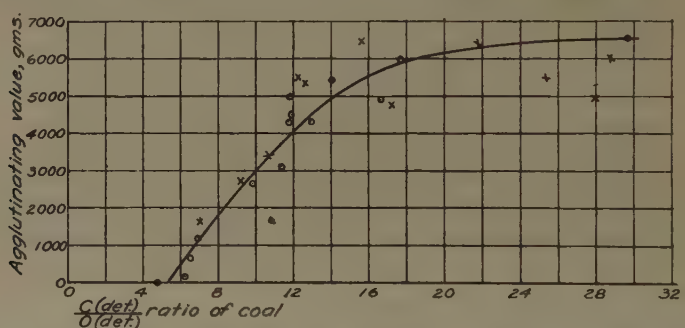


FIG. 14.—RELATIONSHIP BETWEEN AGGLUTINATING VALUE OF COAL AND CARBON-OXYGEN RATIO OF COAL.

kind, while a few coals with ratios as low as 50 coke in the beehive oven, though very rarely producing a good article. The coking property seems to depend not so much on the amount of available hydrogen, which is a very imperfect index of the proportion of the elements in the volatile, as on the relative amount of the hydrogen compared to that of the oxygen. In coals undergoing change to anthracite, the hydrogen-oxygen ratio may fail as a guide; . . ."

The result of correlating the agglutinating values with the $\frac{H}{O}$ ratio is shown in Fig. 15. In general, these ratios seem to bear a somewhat more consistent relationship to the agglutinating values than might have been expected from Rose's diagrams. It is interesting to note that, whereas White regarded 0.59 as the dividing line between coking and noncoking coals using beehive ovens, coal H (Table 3) with a ratio of 0.50 gives a coke in by-product ovens strong enough for blast-furnace use. As the hydrogen content of bituminous coals does not vary greatly from 4 per

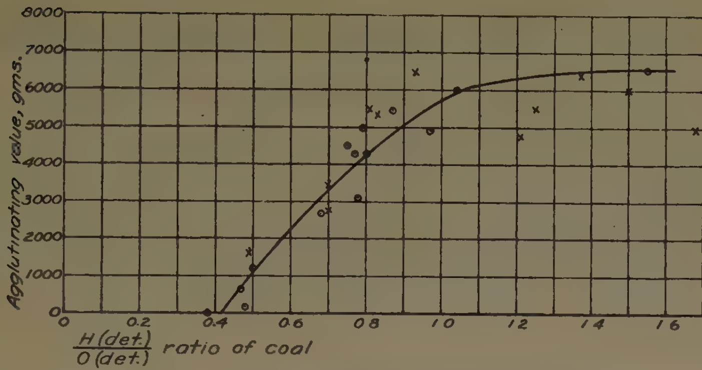


FIG. 15.—RELATIONSHIP BETWEEN AGGLUTINATING VALUE OF COAL AND HYDROGEN-OXYGEN RATIO OF COAL.

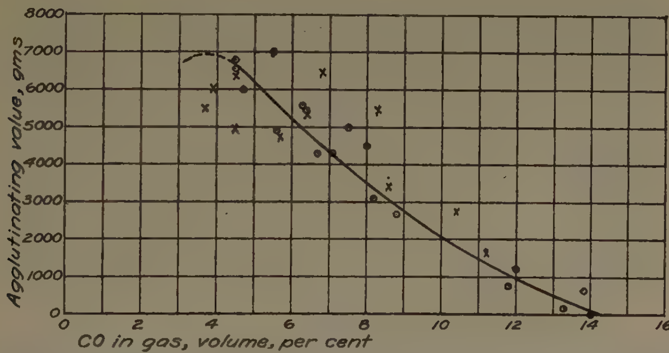


FIG. 16.—RELATIONSHIP BETWEEN AGGLUTINATING VALUE OF COAL AND PERCENTAGE BY VOLUME OF CARBON MONOXIDE IN THE GAS FROM THE DISTILLATION TEST.

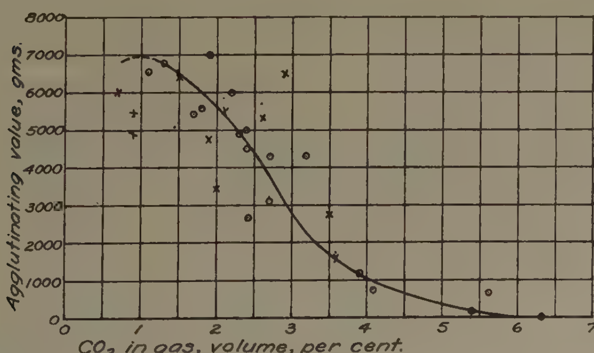


FIG. 17.—RELATIONSHIP BETWEEN AGGLUTINATING VALUE OF COAL AND PERCENTAGE BY VOLUME OF CARBON DIOXIDE IN THE GAS FROM THE DISTILLATION TEST.

cent., the fraction $\frac{H}{O}$ will be almost entirely dependent on the oxygen content, hence the $\frac{H}{O}$ ratio is closely equivalent to the reciprocal of the oxygen percentage and Fig. 15 resembles Fig. 11 when the scales are suitably adjusted.

As Ralston's work[34] seemed to show that the "available hydrogen" as advocated by Parr[31] is an indication of coking strength, this figure has been calculated in column 13 of Tables 3 and 4. No consistent relationship can be discerned between this figure and the agglutinating value.

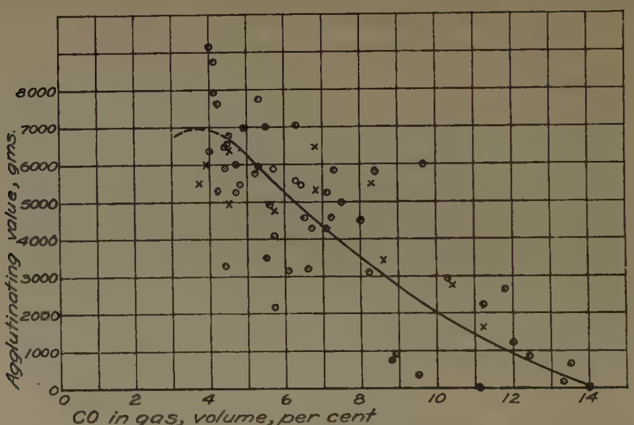


FIG. 18.—RELATIONSHIP OF AGGLUTINATING VALUE OF COAL TO PERCENTAGE BY VOLUME OF CARBON MONOXIDE IN THE GAS FROM THE DISTILLATION TEST, SHOWING MARKED VARIATIONS INTRODUCED BY INCLUSION OF SAMPLES FROM SMALL AREAS.

As a further variation in the study of the relationship of oxygen content to the coking property of the coal, the relationship has been plotted between the agglutinating value and the volume of CO and CO_2 in the gas. The result of these correlations is shown in Figs. 16 and 17. As might be expected, from the previous relationships shown between agglutinating values and oxygen contents the relationships are quite uniform, particularly those involving the CO content of the gas.

Evidently, the latter can be used to indicate in a general way the coking properties of a coal. That this is true has been known for a number of years by investigators of coking coals who have used a distillation train such as has been used by the United States Steel Corpn.[27] for determining the yields of by-products to be expected from coals. Experience has shown that if the CO content of the distillation gas exceeds about 11 per cent. the coal will not coke and these tests have demonstrated that the dividing line is at about this point.

Fig. 18, including a number of small samples, shows the same irregularities manifested by the oxygen relationships in Fig. 13.

Summary

The paper contains a summary of the different tests proposed by other investigators, a discussion of the factors affecting the choice of a procedure for the agglutinating test, an outline of the procedure for making the test, and finally, some correlations of the agglutinating values with the characteristics of oven coke, and with constituents of the coals.

Selected Bibliography

1. B. Ahrens: Ein Weg zur Klärung des Backfähigkeitsproblems von Backkohlen. *Brenn. Chem.* (1924) **5**, 268; *Fuel* (1924) **3**, 453 (Abstract).
2. A. Badarau and F. V. Tideswell: (a) The Caking-power Curves of Coals. *Fuel* (1923) **2**, 61.
   (b) The Caking-power Curves of Coals. *Jnl.* Soc. Chem. Ind. (1923) 537A (abstract).
3. M. Barash: (a) A Contribution to the Study of the Coking of Coals, and a Proposed Standard Method for the Determination of the Agglutinating Value of Coal. *Jnl.* Soc. Chem. Ind. (1926) **45**, 151 T, 176 T.
   (b) Proposed Standard Method for the Determination of the Agglutinating Value of Coal. *Gas Jnl.* (1926) **173**, 276 (Abstract).
   (c) Scientific Control of Carbonisation, with Particular Regard to the Agglutinating Power of Coal. *Fuel* (1927) **6**, 532. Reprint from *Gas Jnl.* (Nov. 9, 1925).
4. A. W. Belden: Metallurgical Coke. U. S. Bureau of Mines *Tech. Paper* 50 (1913).
5. C. Berthelot: Standardization of Methods of Testing Coal, Coke, etc. *Chim. et Ind.* Numéro Spécial (May, 1924) 167.
6. W. A. Bone, A. R. Pearson, E. Sinkinson and W. E. Stockings: Experiments on the Coking Principle of Coals. *Proc.* Roy. Soc. (1922) **100A**, 595; *Fuel* (1922) **1**, 61 (Discussion by R. V. Wheeler).
7. J. T. Burdekin:
   (a) An Investigation of the Caking Power of Coal. Dept. of Scientific and Industrial Research (England), *Bull.* No. 8, (1927).
   (b) *Gas Jnl.* (1927) **178**, 29 (A discussion of *Bull.* 8).
8. L. Campredon:
   (a) Détermination Expérimentale du Pouvoir Agglutinant des Houilles. *Compt. rend.* (1895) **121**, 820.
   (b) The Caking Power of Coals. *Jnl.* Soc. Chem. Ind. (1896) 186. (Abstract of article in *Rev. chim. indus.* 1896, 7, 8.)
9. G. Charpy and M. Godchot: Sur les Conditions de Formation du Coke. *Compt. rend.* (1917) **164**, 906.
10. C. H. Chester: Classification and Testing of Coal. *Gas Jnl.* (1923) **164**, 517.
11. A. W. Coffman and T. E. Layng: A Differential Index of the Coking Power of Coal. *Ind. & Eng. Chem.* (1928) **20**, 165.
12. H. N. Dann: Coking of Coals. *Gas Jnl.* (1926) **173**, 477.
13. J. St. C. Deville: The Agglutinating Value of Coals. *Fuel* (1923) **2**, 296 and 345
14. J. G. De Voogd:
    (a) *Amer. Gas. Jnl.* (1926) **125**, 667 (Abstract).
    (b) *Jnl.* Soc. Chem. Ind. (1927) **46**, 177 B (Abstract).

15. J. T. Dunn: The "Agglutinating Power" of Coals. *Jnl.* Soc. Chem. Ind. (1913) **32**, 397.
16. F. Fischer, H. Broche and J. Strauch: Über die Bestandteile des Steinkohlenbitumens und die Rolle der einzelnen für das Backen und Blähen der Steinkohlen. *Brenn. Chem.* (1925) **6**, 33.
17. G. E. Foxwell:
    (a) The Coking Power of Coals. *Fuels* (1924) **3**, 345.
    (b) The Plastic State of Coal, Parts VII & VIII. The relationship between the plastic curve, the coking index, and the composition of the coal. *Fuel* (1924) **3**, 371.
18. W. H. Fulweiler and T. K. Cleveland: The Agglutinating Value of American Coals. Paper before the American Gas Association Convention, 1927.
19. T. Gray:
    (a) The Determination of the Caking Power of Coal. *Fuel* (1923) **2**, 42.
    (b) *Coll. Guard.* (1923) **125**, 698.
    (c) The Determination of the Caking Power of Coal. *Jnl.* Soc. Chem. Ind. (1923) 341A (Abstract).
20. J. Gross and S. R. Zimmerley: Crushing and Grinding, I.—Surface Measurements of Quartz Particles. *Trans.* A. I. M. E., Milling Methods (1930) 7; II.—The Relation of Measured Surface of Crushed Quartz to Sieve Sizes. *Loc. Cit.*, 27; III.—Relation of Work Input to Surface Produced in Crushing Quartz. *Loc. Cit.*, 35.
21. S. R. Illingworth:
    (a) Coal and Its Carbonisation. *Fuel* (1922) **1**, 17.
    (b) The Coking Power of Coals. *Fuel* (1924) **3**, 345.
22. R. Kattwinkel: The Examination of Coking Coals and Estimation of Their Value. *Fuel* (1926) **5**, 347.
23. D. J. W. Kreulen:
    (a) Einige Bemerkungen zum Backfähigkeitsproblem der Steinkohlen. *Brenn. Chem.* (1924) **5**, 381.
    (b) *Jnl.* Soc. Chem. Ind. (1925) **44**, 160 B (Abstract).
    (c) Bijdrage tot de Studie van het Bakkend Vermogen van Steenkool. *Chem. Weekblad* (1926) **23**, 449.
    (d) Über die Korngrosse von Kohlproben. *Brenn. Chem.* (1924) **5**, 281.
24. R. Lessing:
    (a) A New Apparatus for the Coking Test of Coal. *Jnl.* Soc. Chem. Ind. (1912) **31**, 465.
    (b) A New Apparatus for the Coking Test of Coal. *Fuel* (1923) **2**, 152 and 186; Discussion **2**, 138.
25. C. B. Marson and J. W. Cobb: Influence of the Ash Constituents in the Carbonization and Gasification of Coal, with Special Reference to Nitrogen and Sulfur. Part 1.—Preparation and Preliminary Examination of Special Cokes. *Gas Jnl.* (1925) **171**, 39.
26. G. Martin, E. A. Bowes and J. W. Christelow: Researches on the Theory of Fine Grinding, Part II.—A Method of Accurately Determining Experimentally the Surface of Crushed Sand Particles. *Trans.* Ceramic Society (1925–26) **25**, 51.
27. Methods of the chemists of the United States Steel Corporation for the sampling and analysis of coal, coke and by-products. The Carnegie Steel Company, Bureau of Instructions, Pittsburgh, Pa., 1924.
28. Methods of analysis of coal. Dept. of Scientific and Industrial Research (England).
    (a) Interim report, *Bull.* (1923).
    (b) *Fuel* (1924) **3**, 27 and 53.
    (c) *Bull.* 7 (1927).

29. A. A. Meurice: (a) Quelques notes sur le pouvoir cokéfiant des charbons. *Ann. Min. Belg.* (1914) **19,** 625.
(b) The Coking Capacity of Coals. *Coll. Guard.* (1919), 1559.
(c) Determination of the Agglutinating Power of Coals. *Fuel* (1923) **2,** 305.
30. R. A. Mott:
(a) By-product Coke-oven Practice—XI. *Fuel* (1925) **4,** 463.
(b) The Coking Properties of Coals. *Jnl.* Soc. Chem. Ind. (1926) **45,** 737.
31. S. W. Parr: The Available Hydrogen in Coal. *Jnl.* Am. Chem. Soc. (1907) **29,** 582.
32. Pollard: The Coals of South Wales. *Mem.* Geol. Sur. (1908), 11.
33. S. Qvarfort: A Contribution to the Classification of Coals, with Special Reference to Their Coking Power. *Fuel* (1925) **4,** 154.
34. O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals. U. S. Bureau of Mines *Tech. Paper* 93 (1915).
35. E. Richters: Einfaches Verfahren zur Vergleichenden Bestimmung der Backfähigkeit der Steinkohlen. Dingler's *Polyt. Jnl.* (1870) **195,** 71.
36. H. J. Rose:
(a) Selection of Coals for the Manufacture of Coke. *Fuel* (1926) **5,** 562.
(b) Selection of Coals for the Manufacture of Coke. *Trans.* A. I. M. E. (1926) **74,** 600.
37. F. S. Sinnatt and A. Grounds:
(a) A New Characteristic for Coal: The Agglutinating Power Curve. *Jnl.* Soc. Chem. Ind. (1920) **39,** 83 T.
(b) A New Characteristic for Coal. The Lancashire & Cheshire Coal Research Assn. *Bull.* 8 (1921).
(c) *Fuel* (1924) **3,** 143 (Discussion).
38. A. Weighell:
(a) The Agglutinating Value of Some Durham Coals. *Coll. Guard.* (1921) **122,** 1730 (Abstract).
(b) *Fuel* (1922) **1,** 47 (Abstract).
(c) *Jnl.* Soc. Chem. Ind. (1922) **41,** 17 T.
39. D. White: The Effect of Oxygen in Coal. U. S. Bureau of Mines *Bull.* 29 (1911)

DISCUSSION

A. C. Fieldner, Washington, D. C.—The use of the word "agglutinating" by the authors was largely at my suggestion because I felt that we wanted to keep an open mind and not lead anyone to believe that this test was necessarily a measure of the coking properties of a coal. It measures only its agglutinating power, which we believe, however, has some relation to its coking properties. We will have to obtain considerable correlative data before we can use this test with the greatest of intelligence. Furthermore, the work itself is not completed and these men are going ahead with further research. I believe, however, that the test is of great value in making a preliminary survey in a new field where we know practically nothing. By its use, we can at least arrange coals in the order of their agglutinating values. If we determine these values for mixtures of noncoking and coking coals, we can obtain a preliminary measure to be followed by larger scale tests more nearly duplicating coking conditions.

I am not certain that this is the best measure of agglutinating value. There are many adherents for the type of test in which the inert matter is varied and the maximum amount of such inert matter that can be agglutinated with a given coal determined. I think research by many investigators is necessary before we obtain a final satisfactory test which will be acceptable to and adopted by the American

Society for Testing Materials or other standardizing bodies. I am sure that sooner or later such a test will be adopted. Now, at least, we are indebted to the authors of this paper for following out this method of approach. I hope it will encourage others who are working with different test methods to develop their procedures and present the data to the public.

H. J. Rose, Pittsburgh, Pa.—As Mr. Fieldner points out, this test does not necessarily measure the coking power of a coal, but rather, its cementing properties. It measures the ability of a coal when heated to stick together 10 times its weight of hard, rounded, noncoking sand grains. Within the scope of its limitations, it appears to be a promising test and we are much interested in it.

However, when we coke coal, we do not usually use coal to cement together sand or other noncoking materials. We coke coal as it is. It is true that in most coking coals there are various percentages of noncoking or poorly coking materials, slate, mineral charcoal and other noncoking impurities, but these in general represent only a minor proportion of the whole, and certainly no such a ratio as 10 to 1.

The relation between the agglutinating property of coal and its oxygen content, is certainly most interesting and encouraging. We know that coking properties vary definitely with oxygen content.

I feel that we cannot expect too much in the way of predicting the strength of commercial coke, from these crucible tests. In coke-oven practice, the size of the coal as charged, the moisture content, the cubic foot weight, and other physical variables are among the important factors in determining the size and strength of the coke, which will be produced. In each of these respects the laboratory test is necessarily at wide variance with practical conditions.

That is not a criticism of the present test. It is a general criticism of practically all laboratory methods, and is one of the reasons why we must use laboratory tests with great caution when we attempt to predict the coking property of coal. We may congratulate ourselves that the development of this test is in such competent hands and that the men who are responsible for working with it realize these factors, including the wide gap between laboratory results and plant practice.

What is meant in practice by "strong coke"? In general, we mean a coke which, when once sized, will retain that size without further breaking down. We prefer to have a coke that will not break down much in any stage of handling, but it must at least retain its size once it has gone over the screens.

The yield of individual sizes is one of the most important points in determining the suitability or value of a coal for coking purposes. Certain coals when coked under a particular set of conditions will give a high yield of egg-size coke, or larger. Other cokes will break down to give a large yield of nut size. The hardness of the individual pieces of egg and nut coke may be the same, but the coke giving the egg-size coke is considered much "stronger" because it broke down less on handling.

Shrinkage cracks largely determine the degree to which a coke breaks down and are of very great importance. The size range and yield of sizes are major considerations, while the hardness of the individual pieces is only a minor characteristic in most cases.

J. B. Morrow, Pittsburgh, Pa.—May I ask Mr. Fieldner if any work has been done in testing the whole seam of coal on various fractions; that is, 1.30, 1.40, 1.50, 1.60, 1.70, 1.80, 1.90 and 2.00 specific gravities.

A. C. Fieldner.—Not systematically. There are a few tests of washed coals and unwashed coals given in this paper and, I think, the fractions in the float-and-sink test. They show that the agglutinating values of samples of coal fresh from the face differed greatly from values determined on samples of the same coal that had

been lying around the laboratory for six or eight months; the fresh coal was always stronger than the old coal. Of course, we know that coal oxidizes on exposure. This paper has a few tests of the kind you mention but nothing systematic.

J. B. MORROW.—Our experience shows that in order to determine the exact quality of a coal seam it is necessary to test it on different fractions, 1.30 or 1.40 sp. gr. coal, which is the high quality coal; 1.40 to 1.60 sp. gr., which is the middlings, and 1.60 to 2.00 sp. gr., which is the refuse. In our investigations we have found that organic sulfur is low in the 1.30 to 1.40 sp. gr. coal and considerably higher in the middling product between 1.40 and 1.60 sp. gr. In other words, if the washing operation is carried on at 1.40 sp. gr. considerably more sulfur can be removed.

We also find on certain coals that the clean coal at 1.40 sp. gr. has an ash fusion point of about 2500° F. In other seams, or in the same seam, we find that the coal between 1.40 and 1.60 sp. gr. has a fusion point of the ash of about 2300° F. and that the rock end below 1.60 sp. gr. is approximately the same as the low-gravity coal, that is, 2500° F. This indicates that there is something in the middling products between 1.40 and 1.60 sp. gr. that is decidedly off color from the balance of the coal.

A. C. FIELDNER.—Yes, the authors devote one or two pages to calling attention to the necessity of very careful sampling of coals for the test. They found that samples taken from an upper bench were quite different from those from a lower, and that results on samples from different parts of the mine varied. One must be very careful in sampling the material to be sure that it is representative of what is to be used. This indicates that undoubtedly you would find differences in the several specific gravity constituents.

W. L. REMICK, New York, N. Y.—May a nonchemist ask whether or not the glycerin had any effect on the agglutinating power?

A. C. FIELDNER.—The authors made tests on that point and their conclusions were that it had no effect. At first, they moistened the coal with water to prevent segregation of sand and coal dust, but this gave trouble due to evaporation. After a change to glycerin, they got practically the same agglutinating values. On the other hand, if the amount of glycerin or water is varied, there are variations in results. It is important to use a standard amount of liquid on each sample so as to make the values comparable.

W. H. BLAUVELT, New York, N. Y.—I should like to ask Mr. Fieldner whether he knows whether these tests have been correlated in any way with the reports that F. F. Marquard[5] made about a year ago, regarding the cokeability of different coals; whether there is any attempt to correlate those investigations, or whether they are capable of being correlated, as far as he knows now. I thought Mr. Marquard's paper[3] was a step in the right direction and a very important one, but, of course, he did not go all the way.

A. C. FIELDNER.—I am not sure but I do not believe that this had any relation to those tests. The authors did test some coals for their fusing values and it is possible that one of them is the coal used by Mr. Marquard. Coals are not exactly identified by mines in this paper. I think they got their correlative information where they could, perhaps from the operators of eight or nine coke-oven plants using the same coal. These men would state whether the product from a certain coal was a good coke or a poor one, a strong coke or a weak coke, etc.

---

[3] F. F. Marquard: Effect of Coal Segregation, Mixing and Heating on the Quality of Metallurgical Coke. *Blast Furnace & Steel Plant* (1927) 5, 55.

W. H. Blauvelt.—Did their laboratory tests check up with the practical results at these plants?

A. C. Fieldner.—Yes, they checked in that they arranged the cokes in the same order. You might say that the laboratory test will indicate whether the coke is what the trade considers a strong coke, a weak coke, or a medium coke. For example, coal *B* from the Pratt seam, in Alabama, showed one of the highest agglutinating values—6570; it is said to make a strong coke. Coal *H*, from Sunnyside, Utah, gave a value of 1210; this coal is known to give a weak coke. From such results they conclude that we can say, at least, that the agglutinating values give a basis for classifying coals into groups of strong, weak, or medium-coking coals.

H. J. Rose.—I had the privilege of talking with Mr. Bird about this paper recently. At that time he said he was sorry that the arrangement under which some of the coal samples were obtained did not permit the disclosure of more detailed information regarding them. In some cases he had complete information. In other cases he did not.

J. R. Campbell, Scottdale, Pa.—It occurred to me that it might be interesting, while we are talking about coking coals, to speak of the rank of the coals, the low volatiles, the medium volatiles and the higher volatiles. Is there any difference in the agglutinating values? We speak of Central Pennsylvania coals, or low-volatile coals, 16 or 17 per cent.; then go to the next rank, 22 or 24 per cent., and then on up to the high-volatile coals. They are all coking coals. Something along that line might be interesting. Do you have anything on that, Mr. Fieldner?

A. C. Fieldner.—There are no curves showing the relation of volatile matter to the agglutinating values given in the paper.

J. R. Campbell.—Do you not think it is important?

A. C. Fieldner.—There are several Pocahontas coals that were run by themselves and they are among the coals giving the highest agglutinating value—around 8000. We know, of course, that Pocahontas coal by itself makes a very powerful coke, one so powerful that we do not use it alone for that purpose. I think if volatile matter were plotted against agglutinating value, some relationship would be shown within certain limits.

J. R. Campbell.—I had the pleasure of listening recently to a very interesting talk by M. R. Campbell, in which he spoke of the high-rank coals, and in the high-rank coals he put the low-volatile coals; so I infer that they have good agglutinating values. They are high-rank coking coals.

A. C. Fieldner.—It would really necessarily follow from those curves, because with the increase of volatile matter from the Pocahontas up to, for example, the Pittsburgh coal, there is a continuous increase of oxygen. In that field, the agglutinating value decreases as the oxygen increases. Therefore as the volatile matter increases in that range, the agglutinating value will decrease.

H. J. Rose.—When American coking coals are studied and arranged in order of increasing volatile-matter content, it will be found empirically, that speaking in a very broad way, the strength of coke has a general relationship to the volatile-matter content, until you reach a volatile-matter content exceeding say 30 per cent., when, as has been previously shown, coals of the same volatile-matter content will be found which are quite different chemically. For example, a 34 per cent. volatile-matter Pittsburgh seam coal and a 34 per cent. volatile-matter Illinois coal are very different in coking behavior.

J. R. CAMPBELL.—Due to the oxygen content?

H. J. ROSE.—We presume so. At least, it is related to the oxygen content. We also have to make the reservation that the coals must not have been oxidized or weathered, since oxidation greatly affects coking properties without causing a large change in volatile-matter content.

J. R. CAMPBELL.—A few days ago Mr. Rose made the interesting observation that there were some coals in which a slight amount of oxidation actually helped the coking properties. Is that correct?

H. J. ROSE.—Certain coals when coked alone make a poor coke because of excess coking properties. The size and strength of the coke may sometimes be increased by slight oxidation of the coal. It has sometimes been reported that stored coal has made better blast-furnace coke than fresh coal. On the other hand, certain coals, which when freshly mined are somewhat deficient in coking property, will deteriorate upon oxidation no matter how slight. In other words, a coal may lie on either side of the most desirable condition. Oxidation may move the physical properties in the right direction, or it may move them in the wrong direction.

J. R. CAMPBELL.—M. R. Campbell, in his recent remarks, made the startling observation that the reserve coking coals of the country were being rapidly depleted, that is, the high-rank coals, and that it would be only a few years, figuratively speaking, until they were completely exhausted, and that it would be up to us to cast about for methods of using the lower rank coals, which are not strongly coking coals. He went into that very fully. I may say here that there are a lot of potential coking coals in central and western Pennsylvania. They are the natural coking coals of the country. The best coals ever laid down are in those districts where the mountain foldings give these high-rank, low-volatile coals. In other words, the "water has been squeezed out."

S. M. MARSHALL and B. M. BIRD (written discussion).—The authors recognize that the data in their paper are preliminary and that a better correlation must be established with practical measurements of coke strength before the test can be considered reliable enough for general use.

Mr. Rose's definition of "strong coke" is interesting but it does not provide a tangible means for measuring the strength and some measurement is needed for comparison with the laboratory figures which have a definite value in grams. So far as the authors are aware, the only test that is reasonably standardized in the United States is the shatter test. Although the agglutinating values of only a few of the coals have been compared with the shatter tests the agreement between the relative values from different coals is encouraging and with further tests an acceptable relation can be found. Mr. Rose rightly points out that the strength of a coke is much affected by carbonizing conditions; consequently no laboratory test under standard conditions can show a constant relation to all the quality of coke which can be made from a given coal. If the laboratory test is to be of value it must give a constant result from repeated tests.

Mr. Rose points out that the ratio of 10 parts of sand to 1 part of coal is far from the ratio between inert materials and coking materials in a coke-oven charge. This is true and it is possible that this laboratory dilution is too high and that better results can be had with a 5 to 1 ratio or even less. That is one feature of the investigation which must be further studied. Only a few tests were made with other sand ratios, not enough to learn the relation between the strength with the different mixtures for coals of differing inherent coking abilities. The influence of ash is another factor which needs investigation.

In answer to Mr. Morrow's question about the difference between the strength of coals of different specific gravities Table 5 is submitted. The first four tests are from the investigation which form the basis of the original paper while the remaining eight figures are from an investigation which the senior author has just completed on some other coals, all of which were high in oxygen and showed low coking values. The agglutinating values in the two sets of figures are not directly comparable because the earlier tests were made with a different sand than the one which has been standardized by the Bureau of Mines for this work. If the standard sand had been used for the earlier tests the coking index would have been slightly higher but probably not more than 15 per cent. greater.

With the earlier tests the preceding table shows that the agglutinating value is higher for the float 1.50 fraction as compared with the raw coal. There is the same general tendency for the later tests but the change is not so definite, principally because there is less change in the ash content. The irregularity of the last four figures is partly due to the fact that the test on the float 1.70 fraction was made five and six days before the three lighter fractions were tested. These two latter coals drop off rapidly in coking strength with exposure.

J. R. Campbell has asked about the influence of volatile content upon coking strength. The authors have not been able to trace any definite relation between these two features. If the oxygen content be a controlling factor, as the authors believe is probable, then the volatile content can only be a secondary factor as can be seen from the diagram developed by O. C. Ralston.[4] On one plate in his paper Mr. Ralston shows that a coal of given volatile content may have anything between 3 and 16 per cent. or even 3 and 21 per cent. of oxygen. With this wide change there is little alteration in the hydrogen content—the difference being principally in the carbon content.

TABLE 5.—*Comparative Agglutinating Values of Float-and-sink Fractions*

Sample No.	Fraction	Ash, Per Cent.	Agglutinating Value, G.
7–48 Raw	Total	26.13	7051
	Under 1.50 sp. gr.	8.4	9154
10–95 Raw	Total	27.28	926
	Under 1.50 sp. gr.	10.5	3176
SA	Under 1.30 sp. gr.	3.44	4238
	Under 1.40 sp. gr.	5.41	3629
	Under 1.50 sp. gr.	6.53	3730
	Under 1.70 sp. gr.	8.08	3481
RA	Under 1.30 sp. gr.	3.37	3462
	Under 1.40 sp. gr.	5.54	3300
	Under 1.50 sp. gr.	6.89	2740
	Under 1.70 sp. gr.	8.77	2961

[4] O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals. U. S. Bur. Mines *Tech. Paper* 93 (1915).

Loss in Agglutinating Power of Coal Due to Exposure*

By S. M. Marshall,† New York, N. Y., H. F. Yancey,‡ Seattle, Wash., and A. C. Richardson,§ Tuscaloosa, Ala.

(New York Meeting, February, 1930)

A year ago, Marshall and Bird presented a paper in which a new method of measuring the agglutinating power of coal was described.[1] The procedure used was to determine the strength of a button containing 10 parts sand and one part coal, formed by carbonizing the mixture in a crucible in an electric furnace. The result of the determination was reported as the total weight in grams required to bring about complete destruction of the button. Correlations were shown between the agglutinating value and some of the characteristics of the different coals that were investigated. This agglutinating test has been in use fairly continuously during the past five years for coals of widely differing coking qualities, and it has shown reasonably concordant results accurate enough for commercial use, although there are many features of the test which should have further scientific study.

During the past year a study has been made of some coals of low coking strengths, and incidental to the main investigation some data which appear to be of interest and value were obtained on the loss in agglutinating power caused by exposure to the air. Three coals were investigated, all of which were of the Tertiary age; two were from one field in Asia, and one from a field in the United States. The three were similar, as shown by the proximate and ultimate analyses in Table 1 (moisture-free basis). Coals SA and RA generally contain about 5 per cent. moisture and coal P about 3.5 per cent. moisture on the as-received basis. The large difference in the ash contents of the coals, amounting to practically 10 per cent. between SA and RA, and P, makes difficult a direct comparison of their qualities. In the second section of Table 1 the figures have been put upon an ash-free basis.

This table shows that these coals are closely alike. The ultimate analyses are almost identical except for the higher nitrogen content of

---

* This paper represents work done under a cooperative agreement between the U. S. Bureau of Mines and the College of Mines, University of Washington.

Published by permission of the Director, U. S. Bureau of Mines.

† Consulting Engineer.

‡ Acting Supervising Engineer, U. S. Bureau of Mines, Northwest Experiment Station.

§ Assistant Mining Engineer, U. S. Bureau of Mines, Southern Experiment Station.

[1] S. M. Marshall and B. M. Bird: Test for Measuring the Agglutinating Power of Coal. See page 340.

coal RA, which apparently is offset principally by a lower oxygen content. All of the coals contain a large percentage of oxygen and normally would not be considered acceptable for the manufacture of blast—furnace coke. Nevertheless, all three of them have been so used—RA and SA with small additions of a stronger coking coal, and P alone—but the strengths of the cokes are much below the standard that is acceptable in the United States.

TABLE 1.—*Coal Analyses*

Coa	Proximate Analysis			Ultimate Analysis				
	Ash, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	C, Per Cent.	H, Per Cent.	N, Per Cent.	S, Per Cent.	O, Per Cent.
MOISTURE-FREE								
SA	16.11	39.95	43.94	67.50	5.15	1.62	0.84	8.78
RA	15.80	38.39	45.81	68.50	4.99	2.34	0.94	7.34
P	6.02	40.41	53.57	75.88	5.39	1.69	1.07	9.95
MOISTURE-FREE AND ASH-FREE]								
SA		47.63	52.37	80.47	6.14	1.93	1.00	10.46
RA		45.60	54.40	81.35	5.93	2.89	1.11	8.72
P		43.02	56.98	80.73	5.74	1.80	1.14	10.59

As a general rule, coals initially high in oxygen absorb more oxygen readily, and if they initially possess coking qualities, the coking strength is reduced by the exposure. These tests gave an opportunity to learn how the analyses and agglutinating qualities compared. Samples of each coal ground to 40 mesh were exposed in a thin layer for a number of months in the laboratory under conditions much more favorable to oxidation than obtain in practice. The results which follow show that there was an increase in the oxygen contents and a very definite decrease in the agglutinating values due to the exposure.

The results of the coking index measurements are given in Table 2. The intervals shown in all the tables represent the number of days between the first test or analysis and subsequent tests. Coals RA and SA had been out of the mines for about 60 days and coal P for about 20 days prior to the first measurements. The samples were kept in sealed jars from the time they were mined until they were tested; nevertheless some decrease in strength unquestionably occurred prior to the initial test. This is indicated by the results obtained with samples RG and SG. It is probable, however, that the agglutinating value shown by the first tests is not more than 10 per cent. below the strength of freshly mined coal for coals SA and RA and less for coal P.

Tests on Coals SA, RA and P in Table 2 show clearly that there is a

definite and continued reduction in the agglutinating value as the coal is exposed to oxidation. From the analyses, all of the coals seem to be alike, yet RA and SA drop off with far greater rapidity in their coking values than P. This may be due to an increased susceptibility to oxidation, or other deterioration, because of a slight absorption which occurred in the sealed jar during the period of several weeks between mining and testing. This is partly demonstrated by a second series of tests made upon coals SA and RA, but upon separate samples from jars that were sealed for a much longer time before the initial test was made. The interval between zero days in these two tests is 139 days. The results of these second tests are also shown in Table 2.

TABLE 2.—*Agglutinating Values*

Coal	Interval Between Tests, Days	Approximate Time after Mining, Days	Agglutinating Value, Grams	Reduction in Agglutinating Value per Day of Interval, Grams
First Tests:				
SA	0	60	3469	
	54	114	887	47.8
	121	181	488	6.0
RA	0	60	3550	
	59	119	817	46.3
	106	166	606	4.5
	131	191	531	3.0
P	0	20	3729	
	30	50	2810	30.6
	64	84	2475	9.9
Second Tests:				
SG	0	199	2500	
	7	206	894	229.4
	14	213	650	34.9
RG	0	199	2580	
	7	206	989	227.3
	14	213	431	79.7

From these tests it is apparent that during the long interval of 139 days there was a reduction of approximately 1000 g. in the agglutinating value of both of these coals, but that after the jars had been opened and the coals exposed to the atmosphere there was a further drop of over 1500 g. in 7 days, a much more rapid diminution in strength than was shown by the original samples RA and SA in the same period.

No similar study was made of coal P, but from some meager data obtained during the course of the general investigation, the authors formed the opinion that this coal will not deteriorate with such rapidity after storage.

In a previous paper the opinion was expressed that there was a reasonably direct relation between the oxygen content of a coal and its agglutinating value; an effort was made therefore during this later research to learn whether the decrease in agglutinating value could be accounted for by an increase in the oxygen content. Only two ultimate analyses were made of each coal from which the change in oxygen content could be learned, consequently only general deductions can be drawn.

In Table 3, are given the original analyses (tabulated earlier), together with those made after intervals of 253 days for coals RA and SA and of 178 days for coal P. The outstanding changes are: Reductions in the volatile contents, which range from 0.5 to 2 per cent.; reductions in carbon, which for coals RA and SA amount to 2.5 to 3 per cent. and for coal P to about 1 per cent.; and increases in oxygen, which for coals RA and SA amount to 3.46 per cent. and 2.62 per cent. and for coal P to 1.06 per cent.

There is a much greater change in the analyses of coals RA and SA, which are from adjacent mines, than in coal P, which came from a totally different field. The difference in deterioration is evidenced also by the lesser reduction in the agglutinating value of coal P, as shown in Table 2.

TABLE 3.—*Moisture-free and Ash-free Basis*

Coal	Interval, Days	Approximate Time after Mining, Days	Proximate Analyses: Volatile Matter, Per Cent.	Proximate Analyses: Fixed Carbon, Per Cent.	Ultimate Analyses: C, Per Cent.	H, Per Cent.	N, Per Cent.	S, Per Cent.	O, Per Cent.
SA	0	60	47.63	52.37	80.47	6.14	1.93	1.00	10.46
	253	313	45.63	54.37	78.03	6.02	1.90	0.97	13.08
RA..........	0	60	45.60	54.40	81.35	5.93	2.89	1.11	8.72
	253	313	44.57	55.43	78.04	5.87	2.82	1.09	12.18
P............	0	20	43.02	56.98	80.73	5.74	1.80	1.14	10.59
	178	198	42.47	57.53	79.89	5.54	1.79	1.13	11.65

As far as the agglutinating value is concerned, the most significant feature of Table 3 is the increase in oxygen content. Only two ultimate analyses were made. Consequently, the variation in the rate at which the oxygen was absorbed is unknown. That the absorption does not occur at a constant rate has been shown by many researches; Davis and Reynolds in *Technical Paper* 409 of the U. S. Bureau of Mines, Bone in his "Coal and Its Scientific Uses," and many other investigators have found that the absorption and retention of oxygen are more rapid during the early period of exposure than during the later period. A straight line does not represent the increase of oxygen with time; the curve rises more rapidly during the first days and tends to flatten out later.

Undoubtedly, the coals studied would have shown the same relation had more ultimate analyses been made.

Table 2 shows this characteristic in the reduction of the agglutinating value. The decrease in the value during the first days is much more rapid than during the later part of the exposure, as the last column indicates. For coals SA and RA the reduction in value per day during the first periods of 54 and 59 days averages 47 g., while the reduction toward the end of the exposure is from 3 to 6 g. per day.

Approximate comparisons were made between the drop in the agglutinating value and the probable increase in oxygen retention, based upon the results of other investigators, and the general forms of the curves were found to be similar. The data are not complete enough to permit final conclusions to be drawn, but the indication appears to be that the rate of loss in agglutinating power follows the rate of increase in the oxygen content of the coal. A further and more complete investigation of this feature is warranted. The agglutinating measurement seems sufficiently reliable to give concordant results for such an investigation.

Summary

The loss in agglutinating value of three high-oxygen coking coals due to oxidation was determined on samples of fine coal exposed to the air at room temperature. The results show that the agglutinating value of such easily oxidized coals decreases rapidly, which agrees in general with the rapid loss in coking quality that the coals tested are known to show on account of storage. The agglutinating test may be conveniently used as a sensitive method to detect and follow oxidation in a coal.

Relation of By-product Coke Ovens to the Natural Gas Supply of the Pittsburgh District

By Harold J. Rose,* Pittsburgh, Pa.

(Pittsburgh Meeting, October, 1926)

The peak of production from the Appalachian natural gas field was apparently reached about 10 years ago, and the annual production from Pennsylvania, West Virginia and Ohio has now dropped to about two-thirds of the amounts obtained in 1916 and 1917. This falling off in production has occurred in spite of vigorous efforts and large expenditures by producing companies to maintain or increase their output. The greatly increased costs of development and handling are reflected in the present average value of natural gas at the point of consumption, which value in the three states referred to has more than doubled in the past 10 years.

While it is true that the natural gas production of the United States has increased during recent years, this increase has come from southwestern states, particularly from Oklahoma, California, Louisiana and Texas, which states are now leaders in the production of this valuable but fleeting resource. Pennsylvania, originally the leading state in natural gas production, occupies sixth place in the latest (1924) U. S. Bureau of Mines statistics. West Virginia, which held first place for 15 years, has dropped to third rank, and Ohio is the seventh state in the order of natural gas production.

Domestic and industrial demands for gas are constantly increasing everywhere in the United States. These growing demands have accentuated the difficulties in the localities with depleted natural gas supplies, and serious shortages have already occurred during winter months in the Pittsburgh district. Some relief has been obtained by placing restrictions on the industrial use of natural gas, and by increasing the price (which tends to reduce consumption). Yet these are but temporary expedients for checking gas consumption, which fail to solve the problem of adequate future supplies of gas.

Gas is an exceedingly valuable and important form of fuel, and serious restrictions on its use cannot be tolerated for long in any community. As the supply of natural gas declines, it must be supplemented and eventually replaced with some form of manufactured gas. Many cities, including Pittsburgh, are already supplementing their natural gas supply with the manufactured product.

---

* Assistant Chief Chemist, The Koppers Co.

Seasonal Variation in Demand for Natural Gas

One of the difficult problems which the natural gas companies face is the extremely seasonal character of the demand for gas, due to the house-heating load. In the city of Pittsburgh, for example, domestic gas consumption during the month of January averages about five times as great as in the month of August. The low summer requirements are met by shutting in the better wells, and using the compressor stations to maintain necessary line pressures. Gas which is not needed is kept in the ground under pressure until winter months, when wells are connected in as needed, and may be intensively "pumped" when the maximum supply is required. Thus natural gas operators possess a simple, flexible and fairly economical way of meeting their extreme load variations, so long as they are able to procure enough natural gas for their peak loads.

However, the extreme seasonal variation in demand for natural gas greatly increases the practical and economic difficulties of using manufactured gas to supplement the supply under present local conditions, since such gas is needed during only a few weeks in the year. The natural gas companies have the alternative of building expensive manufactured gas plants and letting them stand idle more than 80 per cent. of the time, or else operating such plants continuously, and still further reducing the income, during summer months, from their large investment in wells, compressor stations and pipe lines.

All manufactured gases differ widely from natural gas in respect to heating value per cu. ft., air requirement for combustion, specific gravity, etc., and while gas-burning appliances can be adjusted to burn any particular mixture of natural and manufactured gas with entire success, they cannot be adjusted to burn satisfactorily mixtures which fluctuate widely in character from month to month. This is an important limiting factor which must always be kept in mind.

Types of Gas-making Processes

Two general types of gas-making processes are available for consideration. One type, represented by the blue water-gas process, may be operated intermittently, and for this reason it is well adapted to peak load conditions. While the heating value per cu. ft. is very low, being less than 30 per cent. that of natural gas, its flame temperature is higher, and furthermore it is added at times when the effect of its addition is minimized by the enormous volume of natural gas which is being distributed. At Elrama, Pa. (about 20 miles south of Pittsburgh, on the Monongahela River), there is located a blue water-gas and producer-gas plant having a maximum daily send-out of about 30,000,000 cu. ft., of which blue gas represents some 75 to 80 per cent. of the total. This plant is operated

as necessary during winter months, and the output is mixed with natural gas in one of the systems entering the Pittsburgh district. The heating value of the mixed gas is maintained above 900 B.t.u. per cu. ft.

The other principal manufactured gas is coal gas, which can be most satisfactorily produced on a large scale in by-product coke ovens. Coal gas from this source is the cheapest high-grade gas which can be manufactured. It has a flame temperature higher than natural gas and a heating value per cubic feet nearly twice as high as blue water gas. However, coke ovens must operate continuously, and while their surplus gas output can be varied considerably, as will be discussed later, a coke oven gas production which is large enough to be of real value in winter, might under present conditions prove burdensome in summer. For example, if the proportion of coal gas to natural gas became high during summer months, the properties of the mixed gas would be seriously modified. Furthermore, a large production of coal gas during summer months would still further reduce the income from natural gas at that time, although fixed charges and payrolls for the natural gas properties would continue practically unchanged.

Coke ovens, with their comparatively steady production of inexpensive, high grade coal gas, are admirably adapted for supplying base-load gas. They are already used for this purpose in a number of cities, and this use is very rapidly growing. A fortunate coincidence is the fact that the coke produced in the ovens is the best raw material for the manufacture of water gas.

Favorable Location of Pittsburgh District for Manufacture of Cheap Gas

There is perhaps no place in the world which is more favorably located for the production of cheap manufactured gas in unlimited quantities than the Pittsburgh district. The time will undoubtedly come when this area is supplied with coke-oven gas, supplemented whenever necessary with blue water gas enriched with a moderate proportion of natural gas. This time is not yet at hand, and the transition period from the present high-B. t. u., high-specific gravity natural gas occurring in natural underground storage, to manufactured gases having very different characteristics, and which have to be manufactured in the varying quantities needed from day to day, is beset with many technical and economic difficulties.

It is not my intention to discuss such problems. The remainder of the present paper will be confined to a discussion of the possible part which existing by-product coke ovens might play in supplementing the natural gas supply of the Pittsburgh district, and also the important bearing of a new type of coke oven on this problem.

Coke-oven Gas

When a ton of good high-volatile coal of the sort produced in the vicinity of Pittsburgh is coked in modern by-product coke ovens, about 11,000 cu. ft. of coal gas having a heating value of some 585 B.t.u. per cu. ft. is obtained. This gas contains in vapor form about 3½ gal. of "light oil" (a mixture of crude benzene, toluene, xylene and solvent

Fig. 1.—Annual coal-carbonizing capacities of by-product coke-oven plants in Pittsburgh District.

naphtha). Gas companies, particularly those having a high B.t.u. standard to maintain, usually prefer to leave the light oil in the gas, but by-product coke plants operated in connection with steel works, find it profitable to remove this product by a scrubbing process which recovers the light oil in liquid form, and reduces the heating value of the coal gas to about 550 B.t.u. per cubic feet.

It may come as a surprise to those who are unacquainted with the present development of the by-product coking industry of this country,

to learn that coke-oven gas is being produced in quantities of the same order of magnitude as natural gas. Yet such is the case.

For example, the by-product coke ovens of Pennsylvania can annually produce a volume of coal gas greater than the maximum quantity of natural gas ever produced or consumed in that state in a single year! Or, to consider a larger area, in 1924 the total production of natural gas east of the Mississippi River was 360,000,000,000 cu. ft., whereas the production of coke-oven gas in the same area for that year was 530,000,000,000 cubic feet!

Fig. 1 graphically illustrates the annual coal-carbonizing capacities of the by-product coke oven plants located in the vicinity of Pittsburgh, Youngstown, Ohio, and in northern West Virginia. The area shown in this map has been somewhat arbitrarily chosen for the purpose of this paper, and while it shows all of the by-product coke ovens in West Virginia, it includes only a few of the plants in Pennsylvania and Ohio. It will be noted that the coke ovens shown have an annual coal-carbonizing capacity of 23,000,000 tons, which is equivalent to a total coal-gas production of more than 250,000,000,000 cu. ft. per year. The question in which we are at present interested is, how much of this total amount might be made available for supplementing our natural gas supply.

Availability of Coke-oven Gas

All by-product coke ovens are heated with gaseous fuel, and until a few years ago, they were invariably heated with a portion of the coal gas which they produced. At first, practically all of the coal gas obtained was consumed in heating the ovens, but with the introduction of improved heating methods, gas consumption dropped to less than 50 per cent. of the total production. The underfiring requirement has been gradually reduced until in the best modern ovens, only 35 to 40 per cent. of the gas (or its thermal equivalent in other fuel gas) is required. The remaining 50 to 65 per cent. of the gas, which is not used for oven heating, is called "surplus gas."

In the case of merchant coke plants, or merchant blast furnaces operating their own by-product coke ovens, practically all of this "surplus gas" is available for sale. However, a complete steel plant (by which is meant blast furnaces, open-hearth furnaces and rolling mills combined with by-product coke ovens), consumes an enormous amount of gaseous fuel, and it is generally agreed that such a p'ant needs all of the "surplus gas" from the coke ovens, and all of the blast-furnace gas, for its own use. Each plant shown in Fig. 1, with the exception of the one at Fairmont, W. Va., is operated in connection with a steel plant, and it can be safely assumed that substantially all of the surplus gas from these coke-oven plants is being consumed for purposes to which it is well adapted.

The possible assistance which these plants could give in supplementing the natural gas supply of the Pittsburgh district, is therefore limited to the amount of coal gas which would be released if producer gas (or blast-furnace gas) were used for heating those ovens which are constructed to permit the use of such fuel gas.

Use of Low-B.t.u. Gas for Heating Coke Ovens

It has already been stated that until recent years, all coke ovens were heated with a portion of the coal gas which they produced. Coal gas has a high flame temperature, and the ratio of the volume of coal gas to air required for combustion is small, so that only the air needs to be pre-heated in the regenerators. Cold blue water gas can also be used for oven underfiring, but its cost almost prohibits its use for this purpose.

Producer gas (made from small coke and coke breeze) is an excellent and cheap fuel for heating coke ovens. Since the flame temperature is low, and since producer gas and air are required in about equal volumes, it is necessary to preheat both gas and air in the regenerators. This requires special regenerator construction, and furthermore, important problems in flue design are introduced, owing to the large volume of waste gas which has to be handled.

It was early recognized by the company with which I am associated that the time was not far distant when economic conditions in many localities would compel the use of low-B.t.u. gas for heating coke ovens. With this in mind, the combination oven was introduced. The combination oven was a Koppers oven which contained the modifications necessary to permit heating with either coal gas or producer gas. This design represented an important advance in making coke ovens adaptable to changing environment, and from that time on, combination ovens were strongly recommended for new installations.

The first ovens of this type were built in 1912, and the second plant in 1915 since that time a great many combination ovens have been built. The first actual use of producer gas for underfiring coke ovens was at Providence, R. I., in 1919. At the present time a number of coke-oven batteries are being heated with producer gas, with complete satisfaction in each case.

Recent Developments in Combination Coke Oven Design

The problem of the relation of by-product coke ovens to natural gas supply has been considered in detail in a paper prepared jointly by C. J. Ramsburg, F. W. Sperr, Jr. and Joseph Becker, and delivered before the fifteenth annual meeting of the Natural Gas Association of America, at Buffalo, N. Y., in 1920. However, there have been some very important developments in the intervening six years.

Although producer-gas underfiring is successful in ovens of the size and carbonizing capacity which were until recently considered standard, it was apparent that size limitations would soon be reached. For many years there has been a continual trend towards increasing the amount of coal carbonized per oven per day, and this has meant that the flue system has had to carry increasing volumes of waste gas, a condition which was, of course, much accentuated by the use of producer gas as fuel. The maintenance of very low draft differentials throughout the entire flue system is a most important point in by-product coke-oven operation, and a daily carbonizing capacity was soon reached which it was not practical to exceed, since it would have involved difficult problems of regulation and also have necessitated a further increase in the cross-sectional area of the horizontal flue. This structural change would have seriously interfered with the uniformity of heating, and would have weakened the oven structure.

These and other practical problems, which were retarding the development of by-product coke ovens, were sucessfully solved by the introduction of an oven utilizing new principles of design, which has been named after its inventor, Joseph Becker. The details of construction and advantages of this oven have been discussed in a paper[1] read before the American Institute of Mining and Metallurgical Engineers in 1923, and the only feature which needs to be emphasized in the present discussion is the fact that every Becker oven is a combination oven. That is, the inherent design of this oven is such that it can burn any type of fuel from low-grade blast furnace gas to the richest coal gas.

It was formerly necessary, in the case of each new installation, to decide whether to build the standard coke oven, which could be heated only with coal gas, or (on the possibility that 100 per cent. of the coal gas might be needed at some future time) to build combination ovens, which were somewhat more expensive, and were subject to certain limitations in design. The result was that few combination ovens were built for steel plants.

A single type of oven now meets all requirements, and every oven of this new type is a potential producer of some 11,000 cu. ft. of surplus gas from each ton of coal carbonized. Furthermore, large increases in carbonizing capacity have been obtained. Whereas the standard performance for combination ovens of the former type was 18 tons of coal carbonized per oven per day, some of the Becker oven installations are carbonizing more than 30 tons of coal per oven per day in regular operation, and there is no indication that the practical limit in capacity has yet been reached.

---

[1] Joseph VanAckeren: Heat Distribution in New Type Koppers Coke Oven. *Trans.* A. I. M. E. (1923) **69**, 513

Annual Coal-carbonizing Capacity of New Type Oven

The annual carbonizing capacity of the ovens of this new type which are in operation or under construction in the United States, already totals more than 20,000,000 tons of coal. One-half of this capacity, or 10,000,000 tons, is located at the plants shown in Fig. 1. All of the Becker oven plants which have gone into operation in this area are using coal gas for underfiring at the present time, but if producer gas or blast-furnace gas were used for underfiring, at least 4000 cu. ft. of coal gas per ton would be released for other purposes, or a total of 40,000,000,000 cu. ft. per year. Whether or not this additional gas would be available for sale in any particular instance would, of course, depend upon the fuel requirements of the associated steel plants and upon the price which could be obtained for it.

No combination coke ovens of the earlier type were ever built for iron and steel companies in the vicinity of Pittsburgh, so that nearly 13,000,000 tons of coal-carbonizing capacity must continue to be heated with coal gas. Since the introduction of the Becker type oven, more than 95 per cent. of the new carbonizing capacity built in the district has been of this type, which shows what an important bearing this oven has on our future reserves of surplus coke-oven gas.

At Fairmont, W. Va., a battery of 60 Koppers combination coke ovens designed to be underfired with producer gas, was built in 1920. The daily carbonizing capacity is about 1100 tons of coal. The coke-oven gas is added to a large excess of natural gas, which is then distributed in the usual way. None of the gas from this plant enters the vicinity of Pittsburgh.

Use of Coke-oven Gas in Other Cities

Coke-oven gas is already a large factor in the manufactured gas industry, and its use as "city gas" is growing very rapidly. More than 65,000,000,000 cu. ft. of coke-oven gas was distributed through city mains in 1924, and this amount represents more than half of the total coal gas so distributed.

The following is an incomplete list of cities to which coke-oven gas is supplied as a portion of the send-out, or which will use it as soon as coke ovens now under construction are completed:

Ashtabula, Cleveland and Toledo, Ohio; Baltimore, Md.; Battle Creek, Detroit, Jackson and Saginaw, Mich.; Birmingham, Ala.; Boston, Framingham and Lynn, Mass.; New York, Rochester, Troy and Utica, N. Y.; Camden, Jersey City and Newark, N. J.; Chicago, Ill.; Duluth and St. Paul, Minn.; Ft. Wayne, Indianapolis and Terre Haute, Ind.; Milwaukee, Wis.; Providence, R. I.; St. Louis, Mo., and Seattle, Wash.

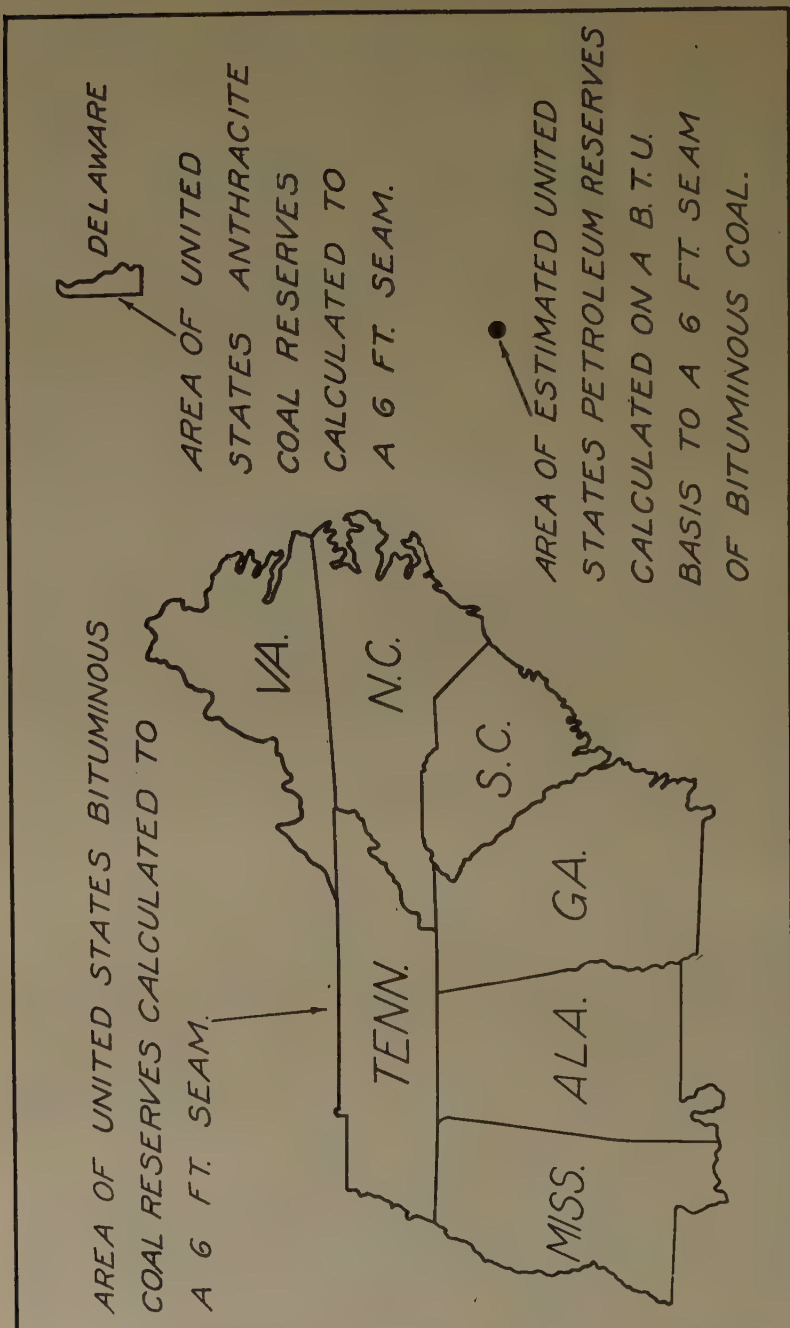


Fig. 2.—Comparative reserves of bituminous coal, anthracite and petroleum in United States.

It has been stated that coke ovens associated with complete steel plants are not likely to have surplus gas for sale, at least under present conditions. At Chicago, Ill., several million cubic feet of gas are obtained daily from such a source, and an enormous additional quantity of gas is obtained from coke ovens not connected with steel plants, in fact the bulk of the gas used in Chicago is produced in coke ovens.

Raw Materials for Gas Manufacture

The three basic raw materials used in the manufacture of gas are bituminous coal, anthracite and oil. Fig. 2, which is based upon the latest and most reliable official estimates, shows our comparative reserves of these three fuels. It clearly indicates that bituminous coal must be the principal raw material for gas manufacture in the future. Since the modern by-product coke oven is the most satisfactory and economical type of coal-carbonizing apparatus, it seems safe to predict that Pittsburgh, located in the world's greatest area of coking-coal production and coke consumption, will eventually depend upon coke ovens for its principal source of gas supply.

Classification of Coal—Introduction

By A. C. Fieldner, Washington, D. C.

Chairman, Sectional Committee on Classification of Coal, American Standards Association

In November, 1926, the American Engineering Standards Committee (now the American Standards Association) called a meeting of representatives of various professional societies and industrial, educational and governmental organizations to consider what action, if any, should be taken on the classification of North American coals. This meeting was called because a system for the use classification of coal, proposed by George H. Ashley, had been referred to the committee by the Coal Mining Institute of America. This meeting, at Pittsburgh, was well attended and was definitely in favor of taking up the whole question of scientific and use classification of coals, including all the various ranks from lignite to anthracite. The American Society for Testing Materials was recommended as the sponsor organization to take charge of the work and organize the sectional committee according to the rules of the American Standards Association.

The American Society for Testing Materials proceeded with the organization of the sectional committee, holding a meeting in Philadelphia, June 10, 1927. Officers were elected, regulations adopted, and general plans for carrying out the work were outlined. The total membership of 28 comprising the sectional committee consists of 8 producers, 2 distributors, 9 consumers and 9 members representing general and scientific interest. From this committee three technical committees of 10 to 12 members each were organized: Scientific Classification, H. J. Rose, chairman; Use Classification, W. H. Fulweiler, chairman; Marketing Practice, F. R. Wadleigh, chairman.

The Technical Committee on Scientific Classification was requested to formulate a system of coal classification based on chemical and physical properties of coal and with reference to origin and constitution. The Technical Committee on Use Classification was charged with developing a system of classification, if possible, based primarily on the uses of coal and commercial practice; this system to be correlated with the scientific system as far as possible and desirable. The Technical Committee on Marketing Practice was formed to collect and collate information on commercial practice for the benefit of the other two committees.

The first meetings of the technical committees were held on Nov. 17, 1927, and the first informal progress reports were submitted to the

sectional committee at the annual meeting on March 29, 1928. Also, at the Annual Meeting of the American Institute of Mining and Metallurgical Engineers in February, 1928, a symposium[1] was held on the classification of coal, at which 13 papers were presented, relating to the classification of coal from different points of view; namely, from the point of view of the chemist, the geologist, the paleobotanist and the various users of coal. These papers were very helpful to the committee in starting its work.

At the last annual meeting of the sectional committee, held March 28, 1929, it was recommended that the progress made by the committee in its first two years, as represented by the work of individual members and subcommittees, should be presented at a symposium on coal classification[2] at the Annual Meeting of the American Institute of Mining and Metallurgical Engineers in February, 1930; also that the manuscripts should be submitted in ample time for preprinting, so that the various papers could be sent out to interested members for discussion at the meeting.

It must be understood that these papers are not official reports of the committee or subcommittees, but are unofficial results of investigations or views of the individual authors. Publication of these papers affords the opportunity for criticism and suggestion. The committee needs to know the point of view of the different industries concerned in the production and use of coal, as well as that of the geologist and chemist.

---

[1] Two of the papers, "The Classification of Coal," by Clarence A. Seyler, and "Pure Coal as a Basis for Classification," by F. V. Tideswell and R. V. Wheeler, were included in *Trans.* A. I. M. E. (1928) **76**, 189, 200. The remaining papers are in this volume, pp. 406 to 488.

[2] See pages 489 to 715.

Classification of Coal from Proximate Analysis and Calorific Value

By W. T. Thom, Jr.,* Princeton, N. J.

(New York Meeting, February, 1928)

Many able men have contributed to the subject of coal classification, and recent publications on the subject have indicated a crystallization of opinion in that connection which promises the development in the near future of a scheme of coal classification which will be of real and increasing value to coal producers and users, as well as to those interested in coal on more exclusively scientific grounds. It is obvious that such a classification must be simple; must be based upon readily ascertainable factors; and if properly developed will apply to the whole range of coals between peat and anthracite and not merely to certain types of coal. Many of the numerous schemes of classification proposed in the past have given quite satisfactory results when applied to the higher rank coals, but have broken down when applied to the younger coals of higher moisture, and, as previously pointed out by the writer,[1] this breakdown has been due to the practice of ruling out "moisture" as an essential component of the volatile matter reported in the customary proximate analysis of coal, whereas a review of the mode of formation and natural history of coal satisfies us that moisture is a normal and natural ingredient among its volatile constituents; and a knowledge of analytical practice shows that the "moisture" and "volatile matter" reported are both mixtures of moisture and volatile matter, rather than sharply differentiated distillation fractions.

After an extended consideration of the subject the writer regards the conventional proximate analyses of coals as affording the information essential for a both practically and scientifically satisfactory classification—assuming that the coal samples compared have been collected and analyzed according to standard methods. Two variables are required for a workable classification, and these appear to be afforded by the calorific value and proportion of volatile constituents (including moisture) credited to the coal on an ash-free basis. The writer originally proposed that the figures used for classification purposes be those given for the "air-dried" and "ash-free" form of the proximate analysis,

* Department of Geology, Princeton University.

[1] W. T. Thom, Jr.: Moisture as a Component of the Volatile Matter of Coal. *Trans.* A. I. M. E. (1925) **71**, 282.

but to simplify analytical procedure it apparently is better to use the "as received" form of analysis converted to an "ash-free" basis. Either

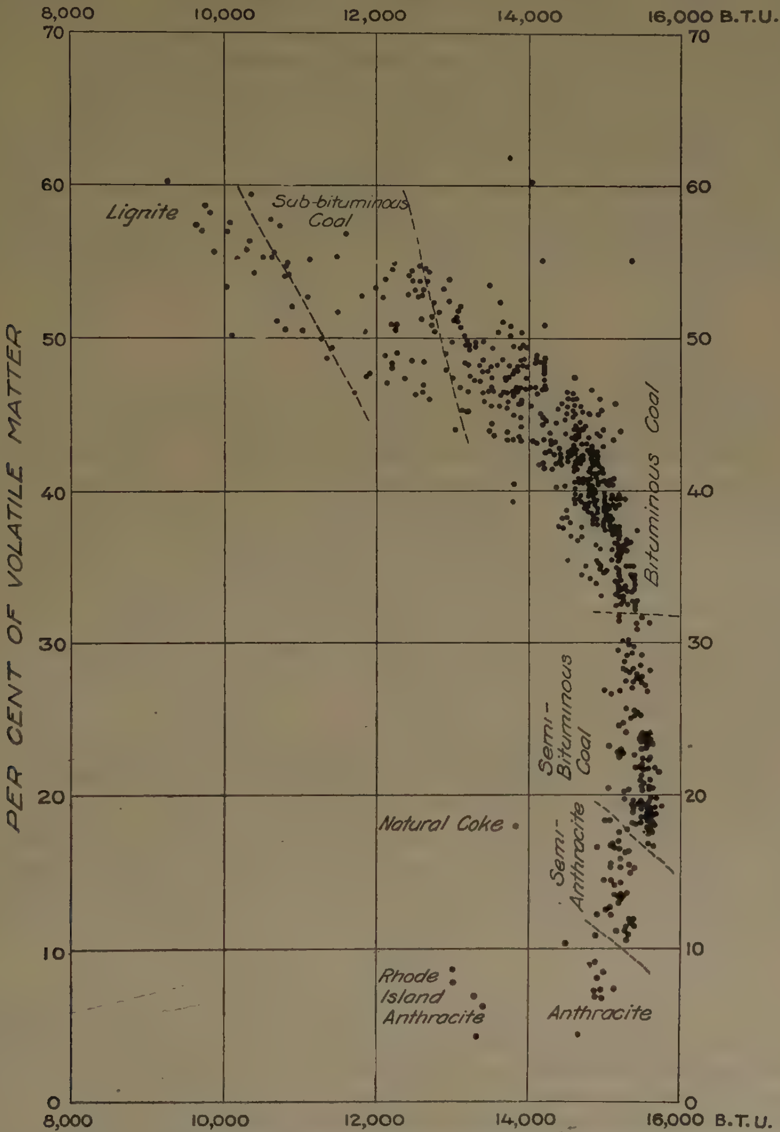


FIG. 1.—VOLATILE MATTER (MOISTURE PLUS VOLATILE MATTER) AND CALORIFIC VALUE OF COALS REPRESENTATIVE OF COAL RANKS AS NOW RECOGNIZED BY THE U. S. GEOLOGICAL SURVEY. ANALYSES PLOTTED ARE ON AIR-DRIED, ASH-FREE BASIS.

scheme of classification promises to give a separation of coals of different chemical, physical and use properties sufficiently clean-cut for practical purposes, as is suggested by Fig. 1.

We all agree, I believe, that in the main nature operates in an orderly and reasonable manner, and consequently that a simple, workable and useful classification of coal is possible. Furthermore, that a satisfactory scheme of classification will show differences between coals of different geologic age reflecting the evolutionary change in plant types which has taken place in the past. It is to be expected that our woody western coals of Cretaceous and Eocene age will show chemical as well as visible physical differences from prevailing Paleozoic coal types, due to changes in the peat-forming vegetation which took place between Permian and Upper Cretaceous time, and that the results given in our chemical analyses will therefore to a degree be related to geologic age as well as to the environmental conditions under which the coals were formed.

Summary

In conclusion, therefore, I should like to stress the idea suggested by Prof. S. W. Parr some years ago, and reiterated by Dr. G. H. Ashley, as to the importance of moisture in the composition and use of coals, and therefore in their classification. Many of the significant physical properties of coals are directly related to their proportions of contained moisture. The remaining physical properties appear to be expressions of chemical differences in the combustible parts of the coal, which are in turn related to and reflected by the differences in the heating values of the different coals shown in proximate analysis. Facts presented by Dr. Thiessen in his paper,[2] will I believe, strengthen and support these general propositions.

Doubtless for a workable scheme of coal classification some general system of nomenclature will have to be developed and agreed upon. And for the reasonably precise determination of use properties from ordinary analyses a graphic classification (on a suitable scale) will, it is believed, be found essential. For this purpose the general form given in Fig. 1 (which is based upon a form proposed by Professor Parr) appears to be most serviceable.

DISCUSSION

M. R. Campbell, Washington, D. C.—I believe I reflect the opinion of some of our leading botanists in questioning the statement that the structure of plants living in Paleozoic times differs much from those living in Mesozoic times; therefore that could not be considered a very important element in determining differences or causing differences. Am I correct in my interpretation? My principal authority is a recent statement by the late F. H. Knowlton.

R. Thiessen, Pittsburgh, Pa.—This is a rather complicated matter and it would take quite a long discussion in order to bring it out thoroughly. It is part of the subject on which I shall speak later but I shall say just a few words now:

---

[2] See p. 419.

We must take into consideration that in the Paleozoic times we had relatively few species of plants to deal with. The chemical composition of all plants, no matter in what age, was much alike; that is, the functions of the living plants were about the same as they are today, but the structure of the Paleozoic plants was different from the structure of the Mesozoic plants as a whole. Take the old Calamites, for example. They were large trees over 100 ft. in height and several feet in diameter; there are records of some that were as much as 6 ft. in diameter. These trees had a shell of wood that was only about 2 in. thick, in which there was a lining of pith, of only a few inches in thickness, and the center was hollow. On the outside of the shell of wood there was again a layer of pithy tissue in turn surrounded by a rather hard tissue. The trees of Mesozoic times were exactly of the same structure as our plants today; they were solid and compact. Coal formation, of course, was largely due to fungal and bacterial decomposition and disintegration, and of course those large shells, put together like staves in a barrel, disintegrated much more readily than would the solid trunks of our pine trees, for example, or oaks.

Again, as far as is known, the Paleozoic plants contained fewer toxic substances, like resin and tannin. Our conifers particularly are full of resinous matter, and this was also true of the Mesozoic conifers; the oaks and similar trees are full of tannins. These are preservative components and tend to preserve the wood more than the woods were preserved in the Paleozoic times.

We must then not only take cognizance of the difference in structure of the Paleozoic plants as compared with our plants of today, but we also must take into consideration the chemical ingredients and their relative amounts in those plants. I have said that the chemical composition was the same as a whole but there is a difference in the amounts of the different chemical compounds.

We must compare, for example, the Calamites with the horsetails. We know that the chemical composition of the horsetails is very different from the chemical composition of a pine tree, for example. I do not mean to say that the lignin and cellulose differed, but in connection with the lignin and cellulose there were other compounds that were in the one and not in the other, or in very different amounts in the one than in the other.

This is a rather long and difficult matter and it would take a long time to discuss it thoroughly.

A. C. Fieldner, Washington, D. C.—Professor Parr bases his system of classification on two factors, namely, the calorific value of the "unit coal" substance and the percentage of "unit coal" volatile matter. These values are plotted on rectangular coordinates, the British thermal unit as ordinates and percentage volatile matter of "unit coal" as abscissas. The different classes of coal group themselves in fairly definite fields, especially the anthracite, semianthracite, semibituminous and bituminous coals. In fact, the eastern type of bituminous coal is separated from the western type of bituminous coal. In the region of subbituminous coal and lignite, the groups are not so clearly marked. This may be due to the samples not representing the coal as it occurs in the mines. As stated in Seyler's paper,[3] Parr's classification can readily be combined with the ultimate analysis method by the scheme developed by Ralston. Parr's method has the advantage of using determinations that are usually made in commercial analyses of coal. This advantage was pointed out by Ralston in his paper[4] published in 1915.

---

[3] C. A. Seyler: The Classification of Coal. Trans A. I. M. E. (1928) **76**, 189.

[4] O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals. U. S. *Bur.* Mines *Tech. Paper* No. 93 (1915).

The method of classification described by Dr. Thom, in which mine moisture and calorific value are used for classification, also possesses the same points of convenience as Professor Parr's system. This method should also be thoroughly investigated by the Classification Committees. It is possible that it can be correlated with the other methods so that finally we will define each rank of coal in terms of all the various limits used in the different systems of classification. For example, in defining low-rank semibituminous coal, we will give the limiting mine moisture and ash-free calorific value as suggested by Dr. Thom; the ash-free calorific value, moisture, volatile matter and calorific value as used by Mr. Campbell in the Geological Survey's method of classification; the limits in unit coal volatile matter and calorific value as required by Parr; and finally plot the carbon, hydrogen and oxygen on a trilinear diagram as suggested by Seyler and Ralston. This system of coal classification will then become a combination and simplification of three or four leading systems now in use.

Classification of Coals by Ultimate Analysis

By H. J. Rose,* Pittsburgh, Pa.

(New York Meeting, February, 1928)

In a paper[1] presented before this Institute in 1926, I briefly discussed the evaluation of coking coals by means of ultimate analysis. The paper contained several graphic studies in which coal analyses calculated free of moisture, ash, sulfur and nitrogen were plotted on triaxial diagrams in which carbon + hydrogen + oxygen = 100 per cent.

From a study of such data for 600 coking coals of the United States, and 150 coking coals of Wales, France and Germany, it was concluded that it was usually possible to predict coking properties and by-product yields with fair success from ultimate analysis. The coals which have proved exceptions to the general rules were found to have physical peculiarities.

Run-of-mine Coal

It must not be forgotten that run-of-mine coal is a composite material consisting of many constituents differing in physical properties and chemical analysis. A coal sample usually contains:

1. Glossy black coal known as anthraxylon or vitrain. This constituent usually has the strongest coking properties, and the lowest ash content.

2. Dull black coal consisting of attritus containing thin layers of anthraxylon. It is also known as clarain. This constituent, in a coking coal, usually has good to excellent coking properties.

3. Mineral charcoal or fusain. This is a porous black material which is soft and crumbly (unless impregnated with mineral matter) and which has absolutely no coking properties. It is different in every respect from the other constituents in coal. The volatile matter content of mineral charcoal is less than 20 per cent. even when it is associated with high-volatile coals. As would be expected the hydrogen content is low, and the carbon content is high.

4. A coal sample may contain tough gray canneloid material which has a high-volatile matter and a high hydrogen content, but is poor in coking qualities.

---

* Assistant Chief Chemist, The Koppers Co.

[1] H. J. Rose: Selection of Coals for the Manufacture of Coke. *Trans.* A. I. M. E. (1926) **74**,600.

5. A coal sample may contain tough gray splint or laminated coal which is not greatly different from the coking constituents in analysis, but which has poor coking qualities.

None of the above constituents have a definite chemical composition. They are physical types whose chemical analysis and general properties depend on the rank of the coal with which they are associated.

Coal also contains visible mineral impurities such as shale, clay, pyrites, etc., which have an effect, sometimes very considerable, on coke structure.

Since coal is a mixture of dissimilar constituents which occur in varying and undetermined proportions, it can not be expected that the average analysis of such a mixture will in every case permit a reliable conclusion to be drawn as to the general character of the coal. For purposes of scientific classification it is hardly satisfactory to say that a coal contains 37 per cent. volatile matter, when in fact it is a mixture of three constituents containing for example 19, 34 and 48 per cent. volatile matter.

A run-of-mine coal may contain such large amounts of mineral charcoal or canneloid material that it resembles coal of another rank in analysis, but not in properties. Or a sample may contain enough splint coal and shale to modify considerably its properties, without very greatly affecting its analysis.

This composite character of coal adds to the difficulty of satisfactory coal classification. Fortunately, extreme cases of the type mentioned above do not seem to be very frequent in actual practice, and they can usually be anticipated if there is an opportunity to examine the coal in bulk. Experience to date does not indicate that ultimate analysis is either more or less useful for detecting these exceptional cases than other characteristics commonly used in coal classification. The systems of coal classification that have been proposed to date do not provide for the fact that coal as produced and used is a mixture of dissimilar materials, yet the composite character of coal has been known for many decades.

Prediction of Coking Properties

In general it may be said that the ultimate analysis of a coal permits its coking properties to be correctly predicted about eight or nine times out of 10. This average can be raised if there is an opportunity to inspect the coal in bulk. The percentages of various constituents present in coal may be roughly estimated by hand separation, but there are at present no satisfactory quantitative methods.

As pointed out by Ralston,[2] the triaxial diagram has mathematical properties which make it valuable in studying the properties of mixtures.

---

[2] O. C. Ralston: Graphic Studies of Ultimate Analyses of Coals. U. S. Bur. Mines *Tech. Paper* No. 93 (1915).

It is also very useful in comparing different systems of coal classification. The paper previously referred to contains a comparison of 150 coking coals of the United States graphed according to their ultimate analysis,

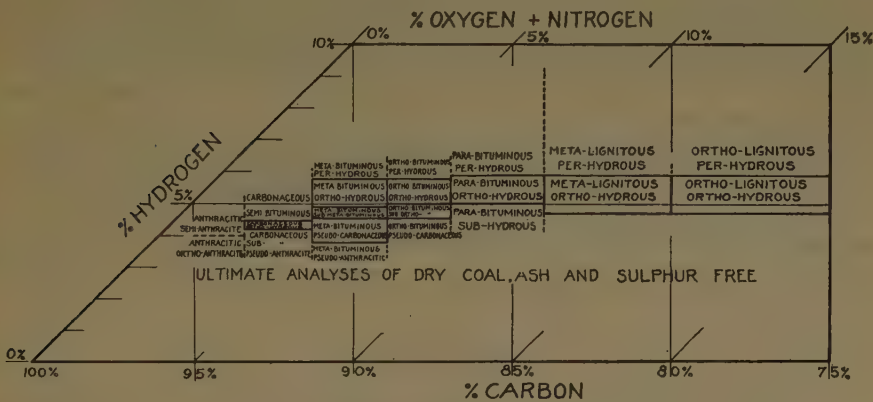


FIG. 1.—SEYLER'S CLASSIFICATION OF COALS.

and by Parr's[3] classification using volatile matter and British thermal units.

Fig. 1 graphically represents Seyler's system of classification[4] superimposed on a triaxial diagram. This classification provides for a wide

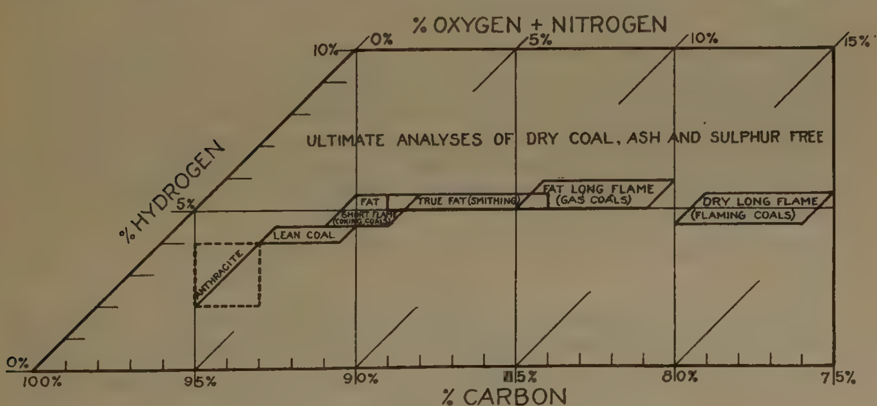


FIG. 2.—GRÜNER'S CLASSIFICATION OF COALS.

range of coal types, and is undoubtedly the most comprehensive system based on ultimate analysis. Seyler has developed a systematic and more or less self-explanatory nomenclature for his classes, but the names

---

[3] S. W. Parr: The Classification of Coal. *Jnl. Ind. & Eng. Chem.* (1922) **14**, 919.

[4] C. A. Seyler: The Chemical Classification of Coal. *Fuel in Science and Practice* (1924) **3**, 15, 41, 79.

are rather formidable, and it is questionable whether they could be popularized.

Fig. 2 presents Grüner's[5] system of coal classification, which is the oldest and perhaps the best known system that is in use today. It is based on ultimate analysis. The overlapping of certain classes and gaps between other classes is well shown by this figure. Grüner's classification, unlike Seyler's, does not provide for coals of unusually high or low hydrogen content.

A detailed comparison of the advantages and disadvantages of these and other systems can best be made by classifying a large number of typical coals of known properties and behavior, by each method, and observing which system of classification best succeeds in separating unlike coals, and in grouping coals of similar properties.

DISCUSSION

G. S. Rice, Washington, D. C.—May I ask what would be the effect if this ultimate analysis is based on air-dried coal?

H. J. Rose.—Since air-dried coal contains moisture, which is composed of hydrogen and oxygen, the position of any coal will be shifted upwards and to the right on the triaxial diagrams, if the moisture remaining in air-dried coal is included in the ultimate analysis. It would result in a greater separation of high-moisture coals on the diagrams.

G. S. Rice.—Does it destroy the value of it for the higher rank coal?

H. J. Rose.—No, I do not think so. I have plotted the ultimate analyses of air-dried coals reported by Dr. Ashley in a paper[6] presented before the Institute, and believe that there are certain advantages in using the air-dry basis. In the studies of coking coals which have been described, only high-rank coals were considered, and there is no objection to comparing such coals on the moisture-free basis. However, I do not see that we can avoid consideration of moisture in a practical classification including lignite and subbituminous coal.

H. N. Eavenson, Pittsburgh, Pa.—Have you enough data to show that on the chart?

H. J. Rose.—The diagram of Dr. Ashley's "type coals" shows how different coals are spaced on the triaxial diagram, when moisture is included in the ultimate analysis.

G. S. Rice.—Is it possible to put into your paper a sheet on which that would be plotted?

H. J. Rose.—I would be glad to do that except that the shape and size of the diagram is such that it is not suitable for reproduction in the Transactions. However, I can furnish the blueprints of the diagram.

M. R. Campbell, Washington, D. C.—I have a rather small point which I would like to mention. Mr. Rose admits that banded coal can not be separated in practice.

---

[5] E. Grüner and G. Bousquet: Classification de la houille. Atlas Général des Houillères. (1911) IIe, 16. Comité Central des Houillères de France, Paris.

[6] G. H. Ashley: A Use Classification of Coal. *Trans.* A. I. M. E. (1920) **63**, 782.

Why then should it be considered in separate parts when you try to classify it? That does not seem to be logical. Of course we have entire beds of splint coal which are of uniform texture throughout but when the coal is banded I can not see the object of separating the bands in an attempt to classify it.

H. J. Rose.—I think that is largely a matter of viewpoint. Chemists like to get at the bottom of things, and if they find two coals of similar ultimate and proximate analysis, but which behave differently, they want to learn the reason. If by picking the coals apart and analyzing the constituent bands, they can explain the difference, that gives them a great deal of satisfaction.

Actually, however, some separation of the different constituents in coal is accomplished whenever it is screened. I have in mind a seam in which a band of splint coal is present. When the coal is screened, the splint coal, which consists of large tough slabs, remains in the largest size, and makes a product which brings a nice price for certain uses. Water-gas men are glad to get this product for generator fuel, coke-oven men are satisfied to have it left out of the product which they coke, and the chemist feels justified in recognizing the existence of the splint band in this coal seam.

When coal is screened, mineral charcoal tends to concentrate in the slack. The natural 60-mesh dust from coal will not coke at all. A considerable proportion of mineral charcoal can be removed from coal by wet or dry-cleaning processes.

The bright and dull bands in coal are often quite different in friability, which results in a corresponding degree of separation when the coals are screened. It is well known that the different screened sizes from certain coal seams vary considerably in analysis and suitability for industrial purposes.

Classification of Coal from the Viewpoint of the Geologist

By M. R. Campbell,* Washington, D. C.

(New York Meeting, February, 1928)

You have just heard several papers on the classification of coal as this subject appears to the chemist; I shall approach it from the point of view of the geologist who, perforce, has to deal with coal as it occurs in the ground and with the miner of coal who knows certain phases of the physical characteristics of coal better than the scientist can possibly know them.

The chemist is familiar with coal as it reaches him in the laboratory, but he seldom has an opportunity to see it in its native condition and to study its physical characteristics and its behavior under varying conditions of weather, transportation and use. He naturally regards the chemical composition of coal as the all-important feature upon which to base a system of classification and he is sometimes inclined to ignore as unscientific any scheme of classification that is based on characteristics other than chemical. The geologist, on the other hand, although he is glad to use the data furnished by the chemist, must depend largely upon the differences that he can see and feel.

Classification Understandable to the Layman

In the very comprehensive program of examination of the coal fields of the West which the U. S. Geological Survey undertook in 1906, it fell to my lot, as the responsible head of the work, to classify the coals in advance of much knowledge of their chemical composition and in regions where mining had not begun. In this work I was forced to devise some sort of classification that would work practically and that would be understandable to the layman in the field. Much of the classification was based on the work of those who had gone before me, but in the low rank coals of the West the field was unoccupied and I was forced to devise a scheme embracing both chemical and physical characteristics. This scheme has stood the test of actual practical application for several years but it needs revision and the sharpening of the limits between the different ranks. I feel that my experience warrants my saying that physical characteristics must be considered and I have faith to believe that some scheme embodying both can be devised.

* Geologist, U. S. Geological Survey.

In making the statements given above I do not wish to be understood as decrying the value of chemical work, but I question whether any classification based entirely on chemical composition will be satisfactory. We do not want a rule-of-thumb classification or one that depends on the arbitrary decision of any person, board or commission; a system to be lasting and secure must have a scientific basis, but does that mean that it must be entirely based on the chemical composition of the coal? Science is not limited to any particular kind of knowledge; it embraces all kinds, and a system based on physical characteristics may be just as truly scientific as one that is based on chemical composition.

If the classification of coal were intended solely for the use of the scientist, then there would be no objection to the use of a complicated system for it is generally supposed that scientists dote on long and complicated formulas and that their greatest delight is to devise something that is wholly unintelligible to the average person. A classification of coal to be successful must be as simple as possible so that it may be understood readily by a person of ordinary intelligence.

I can best illustrate this by an actual case. I am now engaged in preparing a geological report of an important coal field in northwestern Colorado. This when completed will be published by the Geological Survey and will have a world-wide circulation. In this field I have to deal with two ranks of coal, which although very different in their extreme development, merge by imperceptible gradation of one into the other. I must classify these two kinds of coals differently, but how can I make it plain to the operators of the field that I have a rational basis for my separation. The one whose coal falls into the lower rank will naturally feel that he has been unjustly treated and he will and should call on me for an explanation. How shall I proceed to convince him that my decision is just and has a solid foundation on fact? Suppose I show him Parr's or Seyler's charts as my evidence for assigning the coals to different ranks; what would be his reaction? I think you can readily imagine what it would be and that it would not be complimentary to me. I must confess that were I in his place and with his limited understanding of the chemical properties of coal, I should feel much as he does.

Conclusion

In conclusion, I would urge that any scheme of classification of coal to be successful must be simple enough for the average layman to understand and apply, because, if it is not easily understood and applied the mine operators and coal dealers will oppose its adoption and no system however exact and scientifically correct will work unless it finally meets with the approval of the men who are engaged in producing as well as those who are consuming the output of the mines.

I would also urge that no arbitrary scheme of subdivision or of grouping of the different varieties of coal will be successful unless it follows some natural order; and decidedly the most important natural order is the progressive change that takes place after the vegetal material has accumulated in the swamp in which it once grew. The subsequent changes are caused by the weight of the superincumbent rocks that were deposited on the vegetal mass, the pressures exerted within the crust of the earth which result in the formation of rock folds or in the overthrusting of great masses of strata for distances that in some cases are measured by miles, and the interior heat of the earth's crust together with the heat that may have been generated by the thrusts spoken of above. These processes and conditions will sooner or later affect every bed of coal, changing it gradually into higher and higher forms until, in extreme cases, it reaches the stage of anthracite or even that of graphite, or the diamond.

The advantage of such a system as a basis for the classification of coals is that each group or rank is related to the group below from which it developed and to the one above to which it will be changed on further metamorphism. All ranks are interrelated and part of a continuous chain in which there is no break.

DISCUSSION

W. Francis, Washington, D. C.—Mr. Campbell's remarks endorsed my own when I said that the geological conditions must be considered in any classification. I have had experience in America with some of the coals in the West which Mr. Campbell describes. I certainly do think there is a radical difference between those earlier coals and some of the coals which come from the West.

Mr. Campbell talked about pressure and how it was applied and when it was applied. I believe that there is where the difference lies between old and new coals. Consider the formation of coal from wood. First of all, there is what we call a peaty fermentation, namely, the action of the bacteria which decompose the woody material. In many cases—in peat, for instance—one is able to recognize the products of decomposition of the wood, which appear as the humus of the botanist. The question is, is that humus always altered in the same way during the second stage when a gradual transition occurs in the material, and it is transformed from peat through various stages finally to anthracite due to the influence of time and temperature?

Also, suppose that the peaty fermentation has not reached completion before a rise of temperature, due to earth movements, occurred. The mixture of partly decayed wood and decomposition products is subjected to a progressive carbonization. Is the extent of carbonization going to be the same and is the final product going to be the same as when the whole of the wood became "humus" before those temperature changes? I believe that there lies one difference between coals of different ages.

I examined some coals from the West by the methods used in the U. S. Geological Survey classification and found their fuel ratios. Coals from the Price field in Utah contained a high resin content, which increased the volatile content and gave a fuel ratio of 1.1. But those coals are radically different in properties from the coals from the Pittsburgh seam of the same fuel ratio—there is no doubt of that. This is a point which needs emphasis. The proximate analysis is a less reliable guide to the properties of coal than is its ultimate analysis.

Classification of Coal from the Viewpoint of the Paleobotanist*

By Reinhardt Thiessen,† Pittsburgh, Pa.

(New York Meeting, February, 1928)

The question whether the kind, rank and grade of coal is in any way determined by the kind or type of plant from which it originated has been a problem since coal was first studied. Some investigators claim that the kind of plants has contributed none of the attributes that characterize the differences found in the various coals; others claim that all differences are caused by the contributing plant; and others hold that only a few characteristics are inherited from the kind of plant from which the coal originated. Most coalologists, however, give this matter no thought.

It can easily be shown that the kind of plant which contributed to a coal had a marked influence on its kind and type. A number of factors enter into this problem, and all unrelated ones must be carefully eliminated, including metamorphism caused by the pressure of the cover, earth movements, mountain building and higher temperature. Only the fundamental substance—the plants out of which coal was formed—and the causative conditions that gave rise to a peat swamp and the conditions imposed upon the plant and the plant substances during the formation of the peat swamp can be considered. Everything else being equal, the question then is, how far do different classes and types of coal owe their differences to different kinds or types of plants?

Geologic Age of Coal

Coal has been laid down in almost every geological period since the Devonian era. Even as early as the middle Devonian some coal was formed; although of no value it has great scientific significance. The most valuable coal was deposited during the Carboniferous. The coals of England were formed during the Mississippian, and most of the valuable coals of America, Europe and China during the Mississippian and chiefly the Pennsylvanian. From the Permian come the coals of France, Saxony, Thuringia and Schwarzwald in Germany and of Bohemia and of the Ural districts of Russia. From the Mesozoic come the coals

* Published by permission of the Director, U. S. Bureau of Mines.

† Research Chemist and Microscopist, Pittsburgh Experiment Station.

in the Triassic of North Carolina and Virginia, also some in Germany, Sweden, South Africa and Australia. Jurassic coal is found in large workable quantities in Hungary, Persia, Turkestan, Siberia, farther India, China, Japan, Australia and New Zealand. The Cretaceous was the second most important coal-forming era. The lower Cretaceous contains coals locally in British Columbia and in Alaska. The largest quantities of coal in western North America are in the Upper Cretaceous. The Cenozoic, particularly the Tertiary, gave rise to vast deposits of coal in America and Europe. From the Eocene of the Tertiary come the coals of the Rocky Mountain states, of Washington and Oregon, and of south Texas, Louisiana and Arkansas; and brown coals of the Halle and the Weissenfels-Zeitz-Altenburger districts of Germany. The well-known pechkohle of Bavaria are in the Oligocene. In Austria a number of important beds, like the Grazer Bucht, are in the Oligocene. All deposits from the last period of the Quarternary on to the present are considered as peats.

Characteristic Plant Growth

Carboniferous Period

This long geologic record from the Middle Devonian to the present day represents millions of years, and during these ages many different kinds and varieties of coals have been laid down. This period has seen the birth of many orders, families, genera and species of plants, in fact, all of the species of plants of today. At the opening of this period the Middle Devonian records but relatively few kinds. The kinds of plants that prevailed during Carboniferous times also belonged to families totally different from those of today—families now wholly extinct or represented only by dwarfed woody plants, the Lycopodia and small reedlike plants, the horsetails. The representative plants were the calamites, that attained a height of 60 to 100 ft. and more and were 1 to 2 ft. in diameter, with tall slender stems, and many small leaflike branches in whorls. The wood of these stems was a mere cylindrical shell, at the most but a few inches thick, consisting of narrow wedges or sections like the staves in a barrel separated from each other by parenchymatous rays. This woody shell was lined with a layer of pith and surrounded by a layer of pithlike cortex, which in turn was surrounded by a shell of lignified thick-walled cells. These plants were neither resin-bearing nor waxy, but they bore a prodigious number of spores.

Other characteristic plants were the Lepidodendrons, also represented by tall trees, some more than 100 ft. in height and 6 ft. in diameter. Some were unbranched and ended in a bunch of yard-long leaves arranged in vertical rows; others had tall stems topped with a much-branched crown, bearing numerous slender leaves. All bore large cones filled

with enormous numbers of spores, both microspores and megaspores. Although the stems also contained a large pith, they had a more compact cylinder of wood and a thick lignified cortex.

The sphenophylls represented another important group. These had slender ribbed stems, seldom more than ¼ in. in diameter, which bore delicate wedge-shaped leaves in whorls; in some members of the group the leaves were deeply cut and hairlike, indicating a swamp life. Some species must have trailed on other plants; others were little more than herbs, undoubtedly representing the underbrush of the time.

The most highly developed plants of the Carboniferous were the Cordaitales, distantly related to the modern conifers, and with stems and wood closely resembling them. They were tall, slender trees, some more than 100 ft. high, topped with branched crowns, bearing large, lanceolate, leathery leaves.

Finally, there was a group of gymnosperms, both fernlike and cycadlike, the Cycadophytes. They were related to the cycads and similarly organized, but bore leaves resembling those of ferns much more than of cycads. They were seed and pollen-bearing.

Permian Period

The Permian was a period of transition marked with adaptations and eliminations. The Lepidodendron disappeared and the Sigillaria became rare. The Calamites were greatly reduced, and true Equiseta or horsetails appeared in their place. All of the Sphenophylla of the previous period had disappeared and totally new species had appeared. Most of the Cycadophytes had also disappeared, but some new species had appeared. The Cordaites held on in reduced numbers, but new forms of conifers, the Ginkgos, had appeared. Two entirely new types of conifers appeared, the Walchia (resembling the Araucarions and probably their ancestors) and the Volzia (the forerunner of the Sequoias and the bald cypresses).

Triassic Period

The Triassic period showed a remarkable change. It had become distinctly the age of the gymnosperms, the central stage now being held by the cycadlike Bennettitales. The Ginkgos were well represented, Volzia and Walchia had increased in numbers, but all of the gymnosperms were dwarfish. The Cycadophytes were almost gone, and the Sigillarias were represented only by lingering species. The Calamites had entirely given way to true Equisetales, small as compared to the ancient forms, but giants as compared with the living. True ferns now had become abundant, predominantly as tree ferns. The forest appears to have consisted of tree ferns, cycads and primitive conifers.

Jurassic Period

The Jurassic was still the age of gymnosperms. The Bennettitales were in their prime, and cycads had attained high eminence. The Ginkgos also played an important role. True, conifers had slightly modernized, but were not radically changed. They now embrace yews, cypresses, cedars and pines, all ancestral forms. The giant horsetails were still prominent, modern Lycopods were still present, and ferns had become abundant; all of these probably formed the underbrush.

Lower Cretaceous Period

In Europe the Jurassic flora continued well into the Lower Cretaceous; but in America great changes had occurred when these times had arrived. Here angiosperms, both dicotyledons and monocotyledons, had made their appearance. They were all of primitive types, yet all bore definite resemblance to species living today. Genera familiar today, although the species were not modern, were sassafras, laurel, myrica, fig, aralia, oak and eucalyptus. The cycads had dropped to an insignificant place, and the ferns and conifers, while not reduced, were now subordinate to the angiosperms.

Upper Cretaceous Period

During the Upper Cretaceous the whole landscape had attained a modern aspect. More than 90 per cent. of the plants were of the kinds known today. Monocotyledons had assumed greater importance, and grasses had appeared. The Sequoias, cypresses, red cedars, white cedars, pines and firs had attained modern characteristics.

Tertiary Period

At the opening of the Tertiary the vegetation had assumed an appearance very much like that of today. The grasses that had originated in the Upper Cretaceous had developed rapidly and now gave all the vegetation a modern aspect. It is doubtful that Tertiary flora, if reconstructed, could be distinguished from the flora of today.

This brief sketch endeavored to show how species after species, genus after genus, and family after family appeared upon the scene, rose to its height, and then disappeared or sank into the background to give way to new species, new genera, and new families, so that at the present stage of geologic history the plants are wholly different from what they were at the beginning of this resume. It began with a relatively few simple groups, passed through flora of increasing complexities, and now reaches flora of the greatest complexity.

Mosses and Similar Plants

So far the vascular plants, those including and above the ferns, alone have been taken into consideration; nothing has been said of plant genera below the ferns—the mosses, liverworts, lichens, algae, fungi and bacteria. Geologic history tells nothing or very little about them; only a few meager records have been left. We can not, however, assume that all were not represented during the entire coal-forming period. They must have taken some part in the accumulation of coal, and by analogy good proof is at hand to show that they always flourished. Without fungi and bacteria we know that the biologic world could not exist.

The Sphagnum mosses are today recognized as important peat builders. Remains of true mosses are always more or less present in peat. Lichens, liverworts and fungi are always present where moisture prevails and generally form part of peat bog flora, although little of them is recognized in peat itself. Fungi and bacteria are the peat-forming organisms—that is, the transformers of plant substances into peat—and so must have been present at all times, although rarely recognized. Their bodies themselves must have added an appreciable amount to the humins, the chief components of coal.

Effect of Diverse Plants on Diverse Coals

The vital question now is whether the various kinds of plants that followed each other during the geologic periods, or the various plant societies that lived at one time in different localities, were diversified enough in structure or chemical composition to bring about or determine diverse kinds of coal. That they were diversified enough in structure and chemical composition can be shown. In this connection some thought must be given to the chemical composition of plants in relation to decay.

Decay of Plants

Plants Easily Decomposed.—Not all plants, plant parts, or plant products decompose with equal ease. Some plant products, particularly those that take part in living functions or form foods such as sugar, starch, protein and certain fats, decompose very easily through the agency of fungi or of bacteria, or both. The components that form the skeleton or the body of plants vary greatly in this respect. The hexoses, xylans, pentosans and pectins decompose with relative ease. Wood is composed essentially of cellulose and lignin. Of these two, cellulose decomposes rather readily, but lignin is very resistant to decay and when deprived of oxygen hardly decomposes at all. Those plant products that take part in protective coverings and protect against heat, cold, sunshine, water,

air and other gases and chemicals are the most resistant plant products known. Certain waxes, fats, resins, fatty acids and alcohols belong to this group. Spore and pollen exines and cuticles are largely composed of these substances. Also, most waste products, like the resins, terpenes, alkaloids and others are not directly attacked by bacteria or fungi. It is also well known that certain kinds of plants as a whole resist decay much more vigorously than others. Algae, fungi, lichens, liverworts and many mosses decompose and disintegrate very readily. Herbaceous plants, many grasses, and sedges, particularly those that live in aquatic habitats and are of spongy structure disintegrate easily.

Resistance of Woods Containing Toxic Inclusions.—Woods free from toxic inclusions, such as bass, elm, beech, maple and ash, rot very quickly; those, however, impregnated with certain toxic substances, such as oak and walnut, are much more resistant. Many coniferous woods are notably resistant because of their high resin content. Monocotyledons as a whole are less resistant than dicotyledons, and the latter less so than conifers. Loosely organized tissues like pith, cortex parenchymatous tissues, leaf tissues, and most barks disorganize and disintegrate readily. On the other hand, birch bark impregnated with toxic resinous matter is very resistant. Some plants, such as the myrtle, are covered by a layer of wax, or their tissues are filled with waxy substances which remain after the tissues have disintegrated.

Composition of Peat Bogs

Mosses.—Among the lower forms of plant life there is also a great difference in the resistance of the plants as a whole and of certain products of the plants. Mosses are known to be peat builders. The Sphagnum mosses in particular are well known to have built up the well-known high moors. The flora of the Irish peat bogs contain large amounts of mosses, and microscopic examination of any peat formed from flora in which mosses were present brings to light whole moss plants in good preservation, as well as fragments and macerated parts; in fact, they often constitute an important part of the total mass. Liverworts of a number of species are common in many peat bogs, but their remains have not been detected in the peat of whose flora they formed a part. Yet certain remains must surely form part of the attritus.

Lichens.—Lichens, too, always occur in swamps, particularly in wooded swamps; yet their remains are not found in the deposit derived from a flora in which they were a part. Liverworts as well as lichens appear to be subject to decay and disintegrate easily.

Fungi.—Fungi are also important flora in all peat bogs. Fungi, in fact, are the first and the chief instruments of decay. Their hyphae pervade every dead plant as long as it is not submerged, and dead trees

and logs are often literally covered with the fruiting organisms of fungi whose vegetative hyphae permeate the log or tree itself. But of the fruiting bodies nothing is found in the peat to which they must have been adding something of their decomposition products. Of the hyphae of fungi also relatively very little remains; although the burrows of the hyphae are observed everywhere in a piece of wood rotted in the air, very little of the hyphae themselves has remained. The spores of fungi are frequent occurrences, though on the whole they form a very small part of the bulk of plant remains. Fungi as a whole are readily attacked by bacteria.

Algae.—Wherever pools of water are maintained for any length of time they are the abode of a number of species of green and blue-green algae. Moreover, a number of different algae abound on logs or tree trunks as long as moisture is sustained, but microscopic examination of a peat resulting from flora in which algae abounded reveals relatively very small amounts of algal matter. This fact is easily explained: In the first place, the algal bodies are readily attacked by bacteria; and second, algae consist of only a small part of solid matter, only a fraction of 1 per cent., the remainder being water.

There are, however, certain algae that have been shown to be important coal formers. These are oil-algae, whose cell walls consist of a hydrocarbon or contain a very large proportion of oil. They have many characteristics of the blue-greens and have been found in the salt lagoons of South Australia and neighboring islands, where they give rise to coorongite, a rubberlike material high in oil. They have also been reported from Russia, Lake Bakal, China and Mexico. Coorongite is the peat stage of the boghead coals which are derived from organisms that resemble these oil algae very closely.

It may easily be seen how the kinds and types of plants or their products, as well as their variability and the ease with which they decay, determine the nature of their remains in an accumulation of peat. Because they resist decay and disintegration many substances accumulate faster than others, though often originally present in much smaller amounts.

Remains of Wood the Source of Coal

That the humic substances are the chief coal builders has now been definitely established. It has also been well proved that lignin is the main source of humin. As already noted, wood consists chiefly of lignin, cellulose, some pentosan, and xylan. All but the lignin decompose almost completely on rotting; the lignin also is changed in some way into a substance of unknown chemical composition known as humin. The larger part of most peats consists therefore of humin; with this are

always present other more or most resistant plant parts and products. Under certain extreme conditions, the humin also decomposes; then nothing but the most resistant plant products are left. As a rule these are the waxes, higher fats, oils and resins and may together be considered diluents of the humin. The relative amounts of each of these resistant substances may vary greatly and thus determine the nature of the diluent. Then again, the relative amounts of humins and resistant remains may vary in any proportion and determine the nature of the deposit as a whole.

End Products of Decomposition

The end products may be a woody coal or bright coal, an attrital coal, or mixture of the two, a banded coal; or the attritus may consist largely of opaque matter and there is a layer of durain; or the whole coal may consist of an attritus largely humic and the coal is pseudocannel, or the coal may consist largely of spores, cuticles and other resistant matter, and the coal is cannel; and finally, in certain exceptional cases, the main mass may have been derived from oil algae and the coal is boghead.

So far it has been shown only theoretically that the flora of the bog determined to a large extent the kind of coal formed—in other words, that the kind of plants predominant in the bog, together with the conditions imposed upon them, gave rise to a characteristic coal. This fact can be shown from natural conditions after observations in the field by brief examples of the formation of different types of peat in existence today and by following the coals formed backward through the different periods of geologic history.

Formation of Peat

Peat is either formed in undrained depressions or large, flat, undrained areas where constant moisture is assured by collection of the annual rainfall. Open-water bogs, marshes, low moors and wooded swamps are the result; or peat may be formed on higher ground, even on hills and mountain sides, in moist, cool altitudes where constant dampness is assured by frequent precipitations—these conditions result in the high moor.

The open-water bog is the habitat of a number of aquatic plants, such as algae, elodea, potamogeton, hippurus and water lilies. Into this area are constantly blown and there accumulate the pollens and spores from the flora of the surrounding higher levels. As a rule, aquatic plants have a spongy, loose structure and decompose easily. Circumstances favor thorough decomposition and maceration, so that only the most resistant substances remain to form a fine muck, consisting chiefly of spores, pollens and cuticles and a little humic matter. This is the peat stage of cannel coals.

Plants in Swamps

The marsh gives rise to greatly varying attrital substances, according to the kind of plant societies it fosters. The deposit is largely humic and may contain fragments of wood. Different types may be noted:

1. The characteristic plants may include rushes, water lilies and iris, with the rushes predominating. The resultant peat is a finely macerated muck—the rush peat.
2. If the characteristic plants include arrowroot, marsh trefoil and water lilies, with the arrowroot predominating, a similar muck, yet widely differing in nature, is formed—the caladium peat.
3. The flora may consist chiefly of reeds (Phragmites) and other grasses, so that a fibrous peat—the reed peat—is formed.
4. Sedges may predominate and give rise to the sedge peat.
5. Conditions may permit birches and alders to take possession, resulting in a macerated peat—the birch-alder peat.

Forested Swamps

In the United States the wooded or forested swamp is of wide extent; the Dismal Swamp is a well-known example. There are large areas of these swamps in Minnesota, Wisconsin and Michigan. Again, a variety of types can be observed and named after the predominating tree in it. The most common is the cedar swamp, the white cedar or Thuya being the predominant tree. Others include the tamarack swamp, the black-ash swamp, the cypress swamp and the black-gum swamp.

Peat formed from the wooded swamp is of particular interest, because it appears to be analogous to most of the bituminous coals and to many lignites and subbituminous coals. Each peat deposit that results from the different tree types has a distinct character. The greatest distinction is to be found in peat derived from angiosperms (the ordinary leafy trees) and that from conifers. The former, being an easy prey to decomposition, form well-macerated humic peat; the latter, having a toxic resinous content, are more or less resistant to decay and form a woody peat, that contains well-preserved tree trunks and their fragments.

Moors

Low moors are peculiar to certain European countries. They will not be discussed here, but they are similar to the American marsh types. Some German brown coals of the Miocene and the Eocene are said to be derived largely from flora originating in this habitat.

The high moors are peculiar to cool, damp countries like Ireland, England, Norway and Sweden, where the air is almost always cool and moisture-laden. Heather and mosses, with some herbaceous plants and a

sprinkling of conifers and birches, constitute the flora. The peat is decidedly stratified, indicating frequent changes of conditions and flora. The Irish peat as a whole differs greatly from any American peats.

As already mentioned, in the salt lagoons of South Australia and near-by islands flourishes an oil-alga, a colonial unicellular organism probably akin to the blue-green algae. The cell wall of this organism is composed almost entirely of an oily substance. Coorongite is formed from these organisms.

Effect of Climatic and Geologic Conditions on Peat

Peat deposits rarely consist of one kind of peat alone, as the vegetation varies with climatic as well as the geologic conditions, which may have changed from time to time, each condition leading to specific flora and a characteristic type of peat. In the Swedish peat studies an exact chronology has been obtained for the past 12,000 years from which good records of the flora and resultant peats have been obtained. The state of preservation of the plant material also depends upon various factors, such as period of exposure, climatic conditions, position, etc., so that the same kind of flora does not always produce the same type of peat.

It has been adequately shown that each type of peat and each recognized layer of a peat bed yields specific chemical results as to the amount of free humic acids, humates, lignin, cellulose, soluble matter and insoluble residues. It is therefore evident that recent peat deposits were definitely characterized by the kinds of plants that predominated in their formation, and these principles apply to all deposits of past times. In applying these principles it must be remembered that the flora of today is the most complex of all ages, and that as the geologic scale is descended the flora becomes less and less complex and deals with fewer and fewer species and families.

Brown Coals

In going down the geologic scale, skipping the interglacial peats, the brown coals of Germany and Austria may be considered. There are several distinct types, according to origin—the earthy brown coals, probably formed in the low moors; the ordinary brown coals, formed from angiosperms and wooded swamps; and the woody brown coals, derived from coniferous wooded swamps. Some are found in the Pliocene, others in the Miocene, and the more important ones in the Oligocene; but the most important deposits are in the Eocene, all of the Tertiary. The angiosperms predominated during Tertiary times, and recently it has been shown that they also predominated in most of the Tertiary coal-forming swamps and that the conifers were present only in subordinate numbers. It had been supposed that conifers were the chief contributors

because the identifiable woody remains were invariably those of conifers, characteristically of Sequoias and Taxodia. Recent investigations have shown that whereas conifers furnished almost all of the identifiable woody remains, they supplied but a small part of the total mass. The larger part consists of a finely macerated attritus derived chiefly from angiosperms whose remains in considerable numbers have been found. Palms appear to have been prominent. The angiosperms yielded readily to decay and disintegration, leaving little more than an amorphous muck or attritus; the conifers, on the other hand, wherever present, resisted decay and maceration to a far greater extent, due to their toxic resinous contents, and left a large proportion of better-preserved woody material. In other fields, as in the Volpriesen, the conifers actually did predominate, and the result was a woody coal or lignite, in the sense of the German investigators. Evidence is accumulating to show that different types of brown coals owe their characteristic nature to specific groups or types of plants.

Wax in Brown Coal

Many of the German brown coals, particularly those of the Weissenfels-Zeitz-Altenburger districts, are highly bituminous and are highly valued for their montan-wax extraction. The bituminous content varies from field to field and from place to place in the same field, ranging from 5 or 6 to 70 per cent. or more. If the content is less than about 7 per cent. the coal is termed simply "feuerkohle," if above that and under 50 per cent. it is termed "schwelkohle," and if over 50 per cent. it is termed "pyropissit." The last is now becoming very rare. The bituminous matter in these coals is derived from the higher fats, fatty acids and waxes, rather than from the resins which constitute but a small fraction.

A lignite from Volpriesen is composed largely of coniferous woody matter of the cupressinoxylon type. This coal contains 44.5 per cent. of ether-soluble and about 5 per cent. of ether-insoluble resinous matter.

Lignites

Turning now to the lignites of the Tertiary of America, it is found again that the types of certain coals were determined by the types of plants from which they arose. Only a few cases will be noted.

Texas

The lignites of Eocene times, from Hoyt and Rockdale, Tex., are amorphous coals, with a few anthraxylon or woody inclusions derived from conifers. The flora in the main comprised angiosperms. The flora giving rise to the coals of the Camden field was amenable to a still

higher degree of maceration, giving rise entirely to an amorphous coal, a cannel coal. A large proportion of this coal consists of fern spores, pollens and resinous particles, cuticles and other waxy remains. The lignite from Lester, Ark., is a good example of this type. These coals are similar in composition to the German brown coals but are farther advanced in the coalification process.

North Dakota

Lignites from the Fort Union bed of North Dakota and Montana, as at Lehigh, Wilton and Glendive, are on the whole largely composed of woody remains derived entirely from conifers. Approximately 75 to 85 per cent. of the mass consists of such woody inclusions. The relatively small amount of attritus reveals but little evidence of angiospermous plants.

Coals from Coniferous and Angiospermous Flora

A fair number of coals from the Cretaceous have been examined with similar results. It is very easy to distinguish coals derived from coniferous flora from those from angiospermous flora. The former are always woody, whereas the latter are always more or less amorphous. A number of types may be distinguished among the amorphous coals. Microscopic differences are sometimes very marked. The two well-known coals from Sunnyside, Utah, and from Gebo, Wyo., are good examples showing remarkable difference in character and appearance. The Sunnyside coal contains large proportions of attrital matter of granular nature, with large proportions of resinous particles ranging from microscopic sizes to nut sizes, with some inclusions of anthraxylon, among which coniferous remains are rare. The Gebo coal consists of highly macerated humic matter with relatively few woody inclusions. This coal appears to have originated in a wooded swamp, with trees other than conifers.

A series of coals may be enumerated, with almost all woody matter on the one end to all attrital on the other, but this would merely be an uninteresting catalog of phenomena already described.

During Carboniferous times there were far fewer species, families and orders of plants to give rise to coal, and one should expect fewer varieties of coals because of the differences in plants from which they arose. Because the plants as a whole differed widely from those of Cretaceous and Tertiary times coals of a different nature are to be expected; this expectation is confirmed.

Paleobotany of Carboniferous Coal

Some excellent work has been left by investigators in the field of the Carboniferous coals. In the time of Lesquereux, Fayol, Grand 'Eury,

Goeppert, Dawson, Orton and Rogers much time and thought were devoted to the paleobotany of coal, and it is mainly from the findings of these men that the paleobotany of the coal fields can be studied. Since that time such studies have been greatly neglected and even decried. Goeppert, Grand 'Eury and Fayol who, with Lesquereux, paid particular attention to the kind of plants composing coals, all concluded that a certain typical plant or plants entered into the formation of each coal. Coals were never formed from one plant alone, yet one or more plants predominated. Goeppert called some coals Araucarion coals, others Lepidodendron coals. Grand 'Eury called a coal from Rivi-de-Gier a Stigmaria coal, while a coal from near Saint-Chaumont and Chazotte is formed for the greatest part from Cordaites, which gave it specific characteristics; at Peron-Midi and at Gandillon the coal is composed of Cordaites and ferns. What in Grand 'Eury's times were called ferns were in reality Cycadophytes. At some places near Saint-Etienne, Sigillaria make up most of the coal. Ettinghausen states that coals at Radnitz are formed chiefly from Sigillaria and Stigmaria, with Lepidodendron, Calamites and ferns as unimportant contributors. Geinitz called a coal at Plauen Calamites coal. Fayol learned to distinguish coals formed from Calamodendrons, Cordaites, or Cycadophytes as readily as he could distinguish a pile of beech wood from fir wood. Dawson and Lesquereux made similar statements concerning American coals, but fewer exact data are at hand on the kinds of plants that formed any coal.

Characteristics of Pittsburgh Coal

Coal from the Pittsburgh bed has very specific characteristics that easily distinguish it from any other. What the predominating plant that formed this bed was has not been definitely determined. Microscopic examination shows that the spores in the coal, present in large quantities, are to a very large extent of one species. A number of different species of spores exist, but they form only a small part of the total spore matter, seeming to indicate that the flora consisted of one predominating type. Other constituents as seen in thin sections would indicate the same thing. Exactly the same statement can be made of the Lower Kittanning, the Middle Kittanning and the Brookville coals, as well as of other coals that have been studied microscopically.

Cannel Coals

Earlier investigators, such as Lesquereux, Dawson and Fayol, associated Stigmaria and Lepidodendrons, particularly the former, with cannel coals. It may be possible that certain species of Lepidodendrons were adapted to aquatic habitats or conditions required for the formation of cannel coals. From microscopic examination it is known that the organic

matter in cannel coals consists chiefly of spores and some waxy, cuticular and resinous matter. The spores are of a different species, if not of a different type, from those found in the coals.

Boghead Coals

Boghead coals stand in a class alone. In the writer's mind there is now no doubt that these coals owe their existence to certain species of oil-algae.

Conclusion

Enough evidence has been presented to show that the kinds of plants from which a coal originates have influenced its nature and character. For certain coals this influence may have been very slight and for others very marked; sometimes, moreover, the flora determined the nature of the coal very decidedly. It must always be taken into consideration that many factors have determined the nature, quality and rank of coal, and all not related must be excluded before the characteristics passed down by the kind of plants from which they originated can be weighed.

Summary

1. That the kind of plants contributing to a coal had a marked influence on the kind and type of coal produced is easily shown. Of the many factors involved in coal formation, only those directly related must be taken into consideration, and such factors as earth movement, temperature and pressure must be eliminated.

2. Coal has been laid down in every geologic age from the Devonian to the present. During this long period many different kinds, grades and ranks of coals were formed. Relatively few kinds of plants constituted the flora when this period began. Although the existing plants were already fairly highly developed the flora as a whole was simple. As time passed, more and more orders, families, genera and species appeared and older forms disappeared, until today there is a complex and totally different flora from that of the Devonian or even the Carboniferous period. The representative plants during the Carboniferous were Calamites, Lepidodendrons, Sphenophylls, Cycadophytes and Cordaites. The Permian marked a great transition period, so that in the Triassic great changes are noted. The Lepidodendrons had entirely disappeared, the Sigillaria were represented only by a few lingering species, and the Calamites had given way to true horsetails. The Cycadophytes had yielded to the Bennettitales, and Ginkgos had appeared, together with forerunners of modern conifers, Volzia and Walchia. In short, the age had become an age of gymnosperms. Ferns, mostly tree ferns, were also numerous.

3. The Jurassic remained to be the age of gymnosperms. Among these yews, cypresses, sequoias, cedars and pines were the characteristic trees. The giant horsetails were still numerous. Lycopods and ferns also flourished. The Jurassic, however, also saw the birth of the angiosperms, and by the Lower Cretaceous they were well on their way. Familiar plants were sassafras, laurel, myrtle, fig, aralia, oak, eucalyptus and palm. The cycads had dropped to an insignificant place, and the conifers, while not reduced, were now subordinate to the angiosperms.

4. During the Upper Cretaceous the whole landscape attained a modern aspect. About 90 per cent. of the plants were of the kinds known today. During the Tertiary the vegetation had assumed entirely modern aspects. The grasses that had appeared during the Upper Cretaceous now brought the whole flora up to that of today in appearance.

5. No mention has been made of plants below the vascular plants—that is, the bacteria, fungi, algae, lichens, liverworts and mosses. Very few records have been left in the rocks; but we must assume that all were present during all these periods and must have made some contributions to coal. The mosses, at any rate, are today known to be good peat formers.

6. Plants are much diversified in structure and chemical composition. In this relation, decay through bacteria and fungi is a very important factor. Not all plants, plant parts and plant products are decomposed with equal ease. Those plant products that serve as foods or take part in the living functions are decomposed easily. The components that form the skeleton of plants vary greatly in this respect. The hexoses, pentosans, xylans and pectin decompose with relative ease. Cellulose, which with lignin is the basic part of wood, also is decomposed quite easily; lignin on the other hand, is very resistant, and when deprived of oxygen hardly rots at all. Plant tissues, such as cuticles, spore walls and pollen walls, that form part of the protective covering against weather and chemical reactions, are the most resistant plant parts known. Certain waxes, fats, oils, fatty acids, alcohols and resins form such tissues. Also, most waste products, including resins, terpenes and resenes, are not directly attacked by fungi or bacteria. It is also well known that certain plants as a whole are much more resistant to decay than others. Most algae, fungi, lichens and liverworts decompose and disintegrate easily. Herbaceous plants decompose more easily than woody plants, aquatic plants more easily than land plants; angiosperms more easily than conifers. Woods like the oak and walnut, containing toxic substances, are more resistant than woods without them, like bass wood, beech and maple. Particularly resistant are those conifers rich in resins, such as pine, balsam, cypress and cedar.

7. Among the lower plant forms it is notable that algae, fungi, many mosses and liverworts are easily decomposed and disintegrated.

Some mosses, however, are good peat builders, particularly the Sphagnum mosses.

8. There are certain forms of algae whose cell walls are composed largely of an oil or fat which are very resistant to decay and form the so-called coorongite, the peat stage of boghead coal.

9. Because many substances resist decay they accumulate faster than others, although originally present in much smaller amounts.

10. Lignin is the chief source of humin. Wood, as already noted, consists chiefly of cellulose and lignin. The cellulose decomposes and disappears and the lignin remains and in some way is changed into an unknown substance called humin. Ordinarily, humins form the larger part of peat. At times the humins also largely decompose and leave the most resistant plant products, such as spores, cuticles, pollens, waxes, higher fat, and resins.

11. Theoretically, therefore, the flora determines the nature of the resultant deposits. This fact can be shown from natural conditions in the field.

12. Peat is classified according to the flora from which derived. The open water bog gives rise to sapropel—the peat stage of cannel coal. The marsh gives rise to amorphous and fibrous peats: (*a*) the rush peat; (*b*) the caladium peat; (*c*) the reed peat, or (*d*) the sedge peat. The forested peat gives rise to the humic and woody peats: (*a*) when the flora is composed of angiosperms it is an amorphous humic peat; (*b*) when it is coniferous the peat is very woody.

The peats of Ireland and England were formed under entirely different conditions—for instance, in highlands with a cool and very wet climate—and differ vastly from our peats.

13. Among the Tertiary brown coals nearly all classes analogous to the low-moor or marsh peats and the forested swamp peats occur. The majority of the European brown coals are derived from angiosperms; conifers were subordinate.

Similar observations have been made of the American lignites of Cretaceous as well as of Tertiary times.

The older investigators have made similar observations of the Paleozoic bituminous coals and have called certain coals "Calamite coals," "Lepidodendron coals," and "Cordaite coals."

Bibliography

1. N. Arber: Devonian Floras. Cambridge University Press, 1921.
2. E. W. Berry: Devonian Floras. *Am. Jnl. Science* (1927) **14,** 109.
3. T. C. Chamberlin and R. D. Salisbury: Geology, **1, 2, 3.** Henry Holt & Co., 1907.
4. A. P. Coleman: The Anthraxolite of Sudbury. *Am. Jnl. Science* (1928) **15,** 25.
5. J. W. Dawson: Acadian Geology, 2d. ed. Macmillan Co., London, 1868.

6. F. Fischer: Chemie und Kohle. *Brennstoff-Chemie* (1927) **8**, 226.
7. H. R. Goeppert: Prize Essay, 1848.
8. W. Gothan: Kautschuk in der Braunkohle. *Braunkohle* (1924) **23**, 713.
9. W. Gothan: Zur Sumpfmoornatur der Braunkohle. *Braunkohle* (1924) **23**, 865.
10. W. Gothan: Studien uber die Bildung der Schwelkohle und des Pyropissits. Abhandlungen ans der Braunkohlen u. Kali-Industrie (1925) Heft. 6, Halle (Saale). *Braunkohle* (1924) **23**, 725, 771.
11. W. Gothan: Weitere Untersuchungen über Bildung von Braunkohlenflöezen. *Braunkohle* (1923) **22**, 49.
12. W. Gothan, K. Pietzsh and W. Petraschek: Die Begrenzung der Kohlenarten und die Nomenklatur der Braunkohlen. *Braunkohle* (1927) **26**, 669.
13. A. W. Grabau: A Textbook of Geology, **1**, **2**. D. C. Heath & Co., 1921.
14. R. Kattwinkel: Untersuchungen über Cannel- und Pseudocannel-kohle. *Brennstoff-Chemie* (1927) **8**, 375.
15. R. Kidston and W. H. Lang: Devonian Floras. *Trans.* Roy. Soc. Edinburgh (1917) **51**, Pt. 3, No. 24; (1920) **52**, Pt. 3, Nos. 24 and 26; (1921) **52**, Pt. 3, Nos. 32 and 33.
16. B. Kubart: Einiges über unsere Braunkohle. *Braunkohle* (1923) **22**, 546.
17. B. Kubart: Ist Taxodium distichum oder Sequoia sempervirens Charakterbaum der Deutschen Braunkohle. Feb., 1921, Ber. Deutsch. Botan. Gesell.
18. P. Kukuk: Unsere Kohlen. B. G. Teuber, Leipzig-Berlin, 1920.
19. R. Kräusel: Ist Taxodium distichum oder Sequoia sempervirens Charakterbaum der Deutschen Braunkohle. Sept., 1921, Ber. Deutsch. Botan. Gesell.
20. P. Kukuk and H. Bode: Vorkommen von brauner kohle im Westfälischen Karbon. Glückauf (1927) **63**, 1705.
21. R. Lang: Moortheorie und Braunkohlen, Bildung. *Braunkohle* (1920). **20**, 529.
22. R. Lang: Allachthonie und amorphe, geschichtete Braunkohle. *Braunkohle* (1923) **22**, 485.
23. W. H. Lang: Devonian Floras. *Trans.* Roy. Soc. Edinburgh (1925) **54**, Pt. 2; (1926) **54**, Pt. 3.
24. L. Lesquereux: Geology of Pennsylvania, 1858.
25. H. Marcusson: Die Zusammensetzung der Lignite. *Braunkohle* (1926) **25**, 729.
26. S. Oden and S. Lindberg: Einige Torfanalysen im Lichte neuzeitliche Theorien der Kohlenbildung. *Brennstoff-Chemie* (1927) **7**, 165.
27. S. Oden: Peat and Peat Problems in Sweden. *Fuel* (1925) 505.
28. E. Orton: Stigmaria in Mercer Coal Bed. Ohio Geol. Surv. **5**, 850.
29. K. Patteisky and F. Perjatel: Die Steinkohle als Ergebniss ihres Uhrsprungsstoffes und des Grades seiner Inkohlung. *Glückauf* (1925) **61**, 1585.
30. W. Petrascheck: Kohlengeologie der östereischischen Teilstaaten. Vienna, 1925.
31. W. Petrascheck: Zur Klärung der Begriffe Steinkohle und der Braunkohlenarten. *Braunkohle* (1926) **25**, 761.
32. D. B. Reger: The Tygart Valley Devonian Trees of West Virginia. *Am. Jnl. Science* (1928) **15**, 47.
33. D. H. Scott: Studies in Fossil Botany. Adam & Charles Black, London, 1923.
34. J. J. Stevenson: Formation of Coal Beds. *Proc.* Am. Philos. Soc. (1911) **50**; (1912) **51**; (1913) **52**. Reprint in 1 vol., 1911–1913.

DISCUSSION

M. R. Campbell, Washington, D. C.—I gather from this paper that I was somewhat misinformed about the rapidity of decay of the cellulose of wood. My general impression, although I cannot pose as an expert on the subject, is that in

the swamps (especially the Dismal Swamp, which is probably as well known as any, and which has very acid water), the organic acids are a natural preservative and decay of the cellulose is arrested. Of course, some of the wood decays; because it may not be covered by water at all times. Cypress logs have been found in the Dismal Swamp at a depth of 12 or 14 ft., therefore it seems to me the cellulose cannot be going to pieces very rapidly, especially in view of the fact that we find in our lignite beds of the West logs in almost their original condition.

R. Thiessen.—We must consider two phases of peat formation; one is entirely above ground and the other is covered up, chiefly by water. The greatest change in the transformation of the plants occurs while the plant is still in the air. Here the fungi are the agents that do most of the work.

It has been found from experimentation that within three years most of the cellulose of wood in the air under moist conditions has disappeared.

E. W. Parker, Philadelphia, Pa.—What takes its place and what kind of decomposition takes place? Is it soluble and is it carried off by water?

R. Thiessen.—Yes, fungi attack the cellulose first. The cellulose is completely decomposed into water and carbon dioxide and disappears, as far as is known now, almost completely. In experimentation after three years there was only relatively a very small amount of the cellulose left, but we all know that the dead plants ordinarily in the peat bog are exposed to such conditions a good deal longer than three years.

The lignin which largely remains is transformed into a compound or a substance of unknown composition, generally called humin or humus. A number of organic chemists are now working on its composition. In the first place, we do not yet know what lignin is, and humus, being derived largely from lignin, is also unknown.

We find, however, that there is always some cellulose left. In any bog, for example, we find trees growing, whose roots, for instance, are covered up. We also see some trees fallen over and covered up with peat, but still growing. We also know that fungi can not work under water; they must have air in order to exist. So we are quite sure that lignin and cellulose are still intact in some of the trees that are covered up. Here is a problem—we do not know what is becoming of that cellulose. We know, however, that decomposition by means of bacteria is being carried on while the substance is being covered up by the debris and water and at great depths. Some chemists claim that the cellulose of such wood adds to the humin as well as the lignin; others claim that it does not.

There is something else to take into consideration: Trees like the elm, black ash and spruce have no resinous substance and decompose almost completely on the surface. Trees like the cedars, bald cypress, and certain other conifers are full of resins which form preservative substances of the wood. The wood in those trees is not decomposed easily, and the cellulose remains a long time. Those are the trees of which we find the wood intact in the brown coals and lignites. Yet in a number of trees the cellulose has disappeared completely. What has become of that cellulose is a problem.

Some years ago I published various articles about the activity of bacteria in peat bogs and denials have come from all over the world, saying that it could not be possible for bacteria to work at such depths, particularly under those acid conditions. Last fall (1927) I had an opportunity to go back to Wisconsin and check my previous work. Inoculations were made in the peat bogs to a depth of 29 ft. We also carried about a ton of peat with us to Pittsburgh where the work is being continued in the laboratory. About 50 inoculations from various depths were made in the field and there is not one that did not take. So I find that my assertions made several years ago hold good, and I can prove their correctness.

Various experiments are being carried out to study the behavior of these bacteria. We find that temperature and nitrogen supply are important factors. Every peat sample yet collected when enclosed in a proper receptacle as it came from the deposit has revealed some bacterial activity. When such a sample of peat taken from a certain depth showing but a slow action—as shown by the gases produced—is supplied with a proper mineral culture solution in which nitrogen is available, an active evolution of gas takes place within 48 hr. There is therefore no doubt that bacteria exist at all depths. At places or at times they may be somewhat dormant; but should they at any time be supplied with available nitrogen they will revive quickly. Also, should the temperature—which in a bog is fairly low—be raised, a greater activity would ensue.

The humins formed in peat are in part, at least, actual acids. This was shown several years ago by Sven Oden. At that time I believed in the absorption theory, and thought the phenomenon often ascribed to chemical reaction to be physical—that is, absorption by colloids. Results in the laboratory have shown that the humins are acids and that they are able to combine with minerals, such as calcium, in the bog. We find, therefore, in the deposit, free humic acids and humates. The latter appear to be chiefly calcium humate, although sodium and aluminum humates may also be present. Dr. Walter Fuchs has found the same in the brown coals of Germany. He found that 8 per cent. of the mineral matter was thus combined.

W. H. Fulweiler, Philadelphia, Pa.—Would not the action of the Imhoff tank probably parallel to a certain extent the decomposition that may go on? We have Imhoff tanks that are 35 or 40 ft. deep. We know that there is a very active anaerobic action and there is some decomposition that the bacteria will destroy.

Also, is it not true that when you speak of the fungi you really mean the enzymes connected with the fungi? It seems to me it is a little more correct to put it that way.

R. Thiessen.—Yes, but if I said enzymes someone not conversant with the term might not know what I meant, so I said fungi and bacteria. The enzymes are the catalytic agents.

The Imhoff tank is a very good example except that it always contains the nitrogen supply that is required for life. Bacteria must have both oxygen and nitrogen, nitrogen to build the body of the bacteria, the protoplasm, and oxygen for respiration. These bacteria are always anaerobic, that is, while they can not exist without oxygen, they can not exist in free oxygen of the air. They take the oxygen from the plant compound by decomposing it and take the oxygen as it is liberated and use that for respiration. As long as they have a nitrogen, carbon and oxygen supply, life can be sustained, but just as soon as they are deprived of any one of these they become dormant or die.

W. H. Fulweiler.—In the Imhoff tank there is an alkaline condition also.

R. Thiessen.—As we have a method to determine the hydrogen ion concentration we are working on that and find that all of these bacteria are practically working in an acid medium. We found an acidity between 3 and 5 pH in the peat bogs in Wisconsin. It should be remembered that 7 is neutral, any figure below 7 is acid, and above 7 alkaline. So far, we have found no alkaline conditions.

H. J. Rose, Pittsburgh, Pa.—In a case that came under my observation recently it was found that cellulose (in the form of canvas cloth) disintegrated in several weeks in the activated sludge process for sewage treatment.

The Classification of Coal in the Light of Recent Discoveries with Regard to Its Constitution*

By W. Francis,† Pittsburgh, Pa.

(New York Meeting, February, 1928)

Before attempting to describe the application of recently acquired knowledge to the classification of coal it will be as well to consider the objects at which a scientific classification should aim. Here one can not do better than to follow the example of Clarence Seyler,[1] by quoting from Huxley's "Classification of Animals:"

"By the classification of any series of objects is meant the actual or ideal arrangement together of those which are like and the separation of those which are unlike, the purpose of the arrangement being to facilitate the operation of the mind in clearly conceiving and retaining in the memory the characters of the object in question. Thus there are as many classifications of any series of natural or other bodies as they have properties or relations to one another or to other things. But the statement of the characters of the class . . . is something more than an arbitrary definition . . . it expresses firstly, a generalization based on and constantly verified by very wide experience; and, secondly, a belief arising out of that generalization—in other words, the definition of the class is a statement of a law of correlation or coexistence . . . from which the most important conclusions are deducible."

In that same sense writes John Stuart Mill, in the chapter on classification in his "Logic:"

"The ends of scientific classification are best answered when the objects are formed into groups concerning which a greater number of general propositions can be made, and these properties more important, than could be made respecting any other groups into which the same things could be distributed. The properties, therefore, according to which objects are classified should, if possible, be those which are *causes of many other properties*, or, at any rate, which are sure marks of them."

Now in the case of coal, the properties which have been used from time to time for classification purposes are the ultimate analysis, volatile matter, caking power, calorific power, burning properties, etc. Seyler has shown that there is a general law of correlation or coexistence between the ultimate composition and the other properties. The ultimate composition, therefore, of the natural series of bodies known as coal is the

---

* Published with approval of the Director, U. S. Bureau of Mines, and the Director, Safety in Mines Research Board, Great Britain.

† Senior Assistant, Safety in Mines Research Board, Great Britain.

[1] C. A. Seyler: The Chemical Classification of Coal, Pt. II. *Fuel* (1924) 3, 79.

property which he considered best fulfilled the requirements of a basis of classification as laid down by Mill and Huxley. It is certainly true that this classification is one of the most satisfactory in use at the present time, more particularly for coals of high carbon content.

The reason why classifications have given good results, at any rate in the case of the higher ranks of coal, has been discovered recently as a result of experiments based partly on the action of organic solvents on coal and mainly on the action of oxidizing agents.

Oxidation experiments have demonstrated that in all coals one group of compounds predominates. These are the various members of the family of ulmins, which are formed from interactions between the degradation products of the cell walls and cell contents of the original woody material of the coal-forming plants.[2] Ulmins are present in all coals and usually comprise not less than 80 per cent. by weight of the coal. In normal coals the other organic ingredients may be classed briefly as structured plant remains, including such materials as spore exines and cuticles, highly carbonized particles, such as occur in fusain, and amorphous materials, chiefly resins and hydrocarbons. During the progressive carbonization of coal which occurs under the influence of time and temperature, it is quite obvious that the change in properties of the ulmins will be more noticeable than any change in the other ingredients. Moreover, the very chemical nature of the latter makes them less susceptible to the changes that occur in the reactive ulmins.

Our present knowledge of the chemical differences in the organic portions of normal coals may therefore be summed up in one sentence—coals differ from each other in two respects: (1) In the proportions of ingredients present (the ingredients are the ulmins, the structured plant remains, chiefly cuticles and spore exines, resins and hydrocarbons), the ulmins always predominating; (2) in modifications in the character of these ingredients such as determine their rank.

The ultimate analysis of coals groups these mixtures according to the modifications which have taken place in the ingredient which preponderates, namely, the ulmin. Such a grouping can only be approximate because of the varying proportions of other ingredients present. With coals which differ markedly in proportions of ingredients the nature of the volatile matter varies considerably more than does the ultimate analysis. The determination of rank based on total volatile content can therefore have no true significance.

---

[2] W. Francis and R. V. Wheeler: The Oxidation of Banded Bituminous Coal at Low Temperature. *Jnl.* Chem. Soc. London (1925) **127**, 112; The Properties and Constitution of Coal Ulmins. *Loc. cit.*, **127**, 2236.

W. Francis and R. V. Wheeler: The Spontaneous Combustion of Coal: The Most Readily Oxidizable Constituents of Coal. Safety in Mines Research Board *Paper* No. 28 (1926).

A thoroughly scientific classification should include first of all a separation of the ingredients of coal and then a grouping of each ingredient by means of any one of its characteristic properties, for instance, the ultimate analysis or volatile content. Unfortunately, it is not yet possible to separate the ingredients of coal without submitting it to processes which profoundly modify the character of one or another of the ingredients. Classification by analysis of the separated ingredients is therefore not yet practical. Oxidation processes and the action of organic solvents are together at present the only methods available for the separation of the ulmins from the other ingredients of coal, and though during separation the ulmins are altered, it is possible at the same time to measure the modifications in character of the various members of the ulmin family. These modifications apparently cause an increasing resistance to oxidation as the rank of the ulmin increases. An arrangement of coals in order of increasing resistance of their ulmins to oxidation is therefore a true classification of this series of compounds in the sense of the definitions of Mill and Huxley. It has been demonstrated that the other ingredients of coal, besides being present in much smaller proportions than the ulmins, are much more resistant to oxidation and in this connection, within certain limits, may be considered to be inerts. A system of classification based primarily on the reactivity of the ulmins toward oxidizing influences, therefore, is free from the principal defect of other classifications in that differences in the proportions of ingredients present have no effect on the order of arrangement.

Besides measuring the reactivity of the ulmins by measuring their ease of oxidation it is also possible to estimate the proportion present in a given coal during that measurement by taking advantage of one of the most important effects of atmospheric oxidation at low temperatures, which is that the insoluble ulmins become again soluble in alkaline solutions. This treatment does not cause the inert ingredients to do so, therefore it is possible to make an estimation of the relative proportions of ulmins and inerts present. The proportions of resins and hydrocarbons may be estimated by the solvent action of pyridine and chloroform. By combining these two methods the proportions of the main ingredients of normal coals may be measured. Since each ingredient has distinct properties and therefore exerts a separate influence on the properties of the containing coal the value of this separation will be appreciated. Thus recent discoveries have made possible a rational classification of coals that groups them in order according to the reactivity of their ulmins, with suborders for differing proportions of the other ingredients.

Experimental Methods

This system of classification has now been tried out on a series of American coals of wide range, varying in rank from lignites up to high-

rank semibituminous coals, according to the older systems of classification. The reactivities of the ulmins toward oxidation have been measured in three ways. In the first place oxidation by air under standard conditions was undertaken and the reactivity determined in two ways.

1. The rate of combination with oxygen under standard conditions was determined by calculation from the change of weight of the coals and the weights of the oxides of carbon and water evolved.

2. In a parallel set of experiments the rate of oxidation was measured by the rate of formation of soluble ulmins.

3. In order to quicken the oxidation processes a chemical oxidizing agent was used. The rate of oxidation was obtained by the rate of formation of soluble ulmins. The most convenient oxidizing agent so far tried is dilute nitric acid.

Time does not permit description or discussion of the methods in full, but complete details will be prepared for publication elsewhere in the near future. The essential features of the experimental methods are as follows: The coals were extracted with pyridine and chloroform in an inert atmosphere. The residues were mixed, dried and ground to pass through a 100-mesh standard Tyler sieve and remain on a 200-mesh sieve.

In one experiment 10-g. samples of the prepared coal residues were contained in tubes and dry air was aspirated through. The tubes were heated in an air oven whose temperature was maintained constant at 150° C. by means of a constant boiling jacket of glycerin and water. A bimetallic thermostatic control was also fitted as a precautionary measure. The water and carbon dioxide in the exit gases were estimated by the change in weight of absorption trains, and the carbon monoxide was estimated periodically by analysis. Change in weight of the coals was determined from time to time by weighing the containing tubes. From these data the volume of oxygen which entered into the reactions was calculated and placed on a "unit oxidizable material" basis from figures giving percentage of oxidizable material, obtained by extraction of the oxidized coals with caustic potash solution.

A parallel experiment was performed with another portion of the sieved extracted coals. The object of the experiment was to determine the periodic formation of soluble ulmins and the total percentage of oxidizable material. The coals were placed in shallow layers in trays and heated to 150° C. in an oven controlled as described. From time to time ulmin determinations were made by extracting the coals with a 1 per cent. solution of potassium hydroxide, precipitating the ulmin with hydrochloric acid, filtering, drying and weighing upon tared papers. The experiment was concluded when the percentage ulmin formation reached a maximum. As the reaction proceeded the lower rank coals showed an increasing tendency to form water-soluble products. The total percentage of oxidizable material was therefore considered to be

the difference between 100 and the sum of the organic residue and ash. The weight of residue invariably became constant under the experimental conditions.

The nitric acid experiments were performed as follows: New samples of coal were extracted with pyridine and chloroform. The extracted coal residues were carefully ground so as just to pass through a Tyler standard 150-mesh sieve. One-gram samples of each of the resultant products were boiled 8 hr. with 100 c.c. of dilute nitric acid solutions of gradually increasing concentration from $\frac{1}{12}$ normal to normal. After the excess of acid was filtered off the oxidized coals were extracted in a standard manner with a 1 per cent. potassium hydroxide solution. By drying and weighing the residues the percentages of the oxidizable material which became soluble during each treatment were calculated after the ash contents of the residues and of the original extracted coals had been determined.

During atmospheric oxidation the rate of ulmin formation may be determined directly, since there is little change in weight during oxidation and a comparatively small loss to water-soluble products. When the figures are given as percentages this error is not serious. On the other hand, during nitric acid oxidation there is first of all a comparatively large increase in weight even with low concentrations and during the late stages of oxidation wholesale conversion of the ulmin into water-soluble acids and nitrophenols. It is consequently necessary to determine the amount of soluble matter formed by difference from the weight of the residues.

Altogether 35 or more coals have been examined by one or all of these methods; up to date, 16 have been examined completely by all the methods. The results obtained with these coals are shown in the tables following and are typical of the general trend of all the other results obtained.

Atmospheric Oxidation

In Table 1 will be found the ultimate analyses of the coals used, together with the volumes of oxygen reacting for 1 week of oxidation at 150° C., calculated on 10 g. of ash-free, oxidizable matter, consisting almost entirely of ulmin. Table 2 gives the volumes of oxygen used during successive weeks until each experiment was concluded. Generally, the experiments were finished when maximum solubility of the ulmin was attained.

It is at once apparent from the figures for the first week of oxidation that the coals may be placed in order according to reactivity similar to the order of increasing carbon content, that is, the coals of lowest carbon content are the most reactive toward oxygen; and those of high carbon content are least reactive. The agreement is not strictly correct, first

TABLE 1.—*Summation of Data on Coals*

Coals Used	C, Per Cent.	H, Per Cent.	Air Oxidation: O^2 Used in 1 Week, c.c.	Air Oxidation: Time for One-half Ulmin Formation, Hr.	Nitric Acid Oxidation: Concentration in Grams Acid per 100 c.c. to Give 40 Per Cent. Solubility
Gillette, Wyo.	71.9	5.4	Gillette........3310	Gillette........ 140	Gillette.......0.620
Hotchkiss, Wyo.	73.4	6.0	Milk River.....3080	Hotchkiss	Hotchkiss.....0.640
Milk River, Mont.	73.4	5.3	Hotchkiss......2740	Milk River..... 150	Milk River....0.715
Gantar, Mont.	78.4	5.3	Baileys Mills...2730	Gantar......... 332	Gantar.........0.730
Baileys Mills; Bottom; Pittsburgh	78.9	5.9	Hiawatha......2640	Hiawatha...... 334	Baileys Mills..0.755
Elm Grove; Bottom; Pittsburgh	80.4	5.6	Castlegate.....2590	Castlegate..... 334	Elm Grove....0.760
Hiawatha, Utah	80.7	5.8	Gantar.........2590	Baileys Mills... 390	Hiawatha.....0.850
Castlegate, Utah	81.5	5.5	Sunnyside......2540	Elm Grove..... 410	Castlegate....0.880
Sunnyside, Utah	82.5	5.8	Elm Grove.....2360	Sunnyside...... 417	Sunnyside.....0.900
Monongah; Bottom; Pittsburgh	83.3	5.6	Monongah 2280	Monongah..... 425	Monongah....1.01
Delmont; Middle; Pittsburgh	84.5	6.0	Delmont.......2280	Delmont....... 430	Delmont......1.05
Scotch Hill; Pittsburgh.	85.2	5.2	Revere........1960	Scotch Hill..... 550	Revere.......1.78
Revere; Bottom; Pittsburgh	86.1	5.4	Scotch Hill.....1945	Revere........ 555	Scotch Hill....2.53
Jamison; Middle; Pittsburgh	86.5	5.6	Jamison.......1900	Jamison....... 650	Jamison......2.87
Vogele; Bottom; Pittsburgh	87.4	5.4	Vogele.........1360	Vogele......... 915	Vogele, over..6.3
Ocean; Top; Pittsburgh	89.0	4.7	Ocean......... 790	Ocean.........1800	Ocean, over...6.3 Higher than Vogele

TABLE 2.—*Atmospheric Oxidation of Coals at* 150° *C.*

[Oxygen at N.T. P. per 10 g. Oxidizable Material]

Coa	Time, Weeks: 1	2	3	4	6	7	13
Gillette	3310	4110	4450		5620		
Milk River	3080	3720	4130		4880		
Hotchkiss	2740	3270	3875		4830		
Baileys Mills	2730	3525	4200		5820		
Hiawatha	2640	3440	3910	4350			
Castlegate	2590	3470	4040			5450	
Gantar	2590	3350	3850		4180		
Sunnyside	2540	3270	3770		4640		
Elm Grove	2360	3100	3760		5470		
Monongah	2280	3050	3680		5250		
Delmont	2280	2980	3415		4485		
Revere	1960	2640	3030				
Scotch Hill	1945	2610	3150		4580		
Jamison	1900	2465	2925		3775		
Vogele	1360	1930	2310		3320		
Ocean	790	1200	1600		2880		4230

of all because the ultimate analysis is not reliable as a discriminating agent between coals of very near rank and, second, because some of the lower rank coals behave abnormally. Gantar coal, for instance, although much below Castlegate coal in carbon content and therefore presumably lower in rank, reacts only with the same volume of oxygen during the first week. At the same time, in view of the large experimental errors which may occur because of oxidation during treatment before measurement, particularly in the case of the low-rank coals, it is unwise to speculate upon such exceptions without additional evidence. With higher rank coals

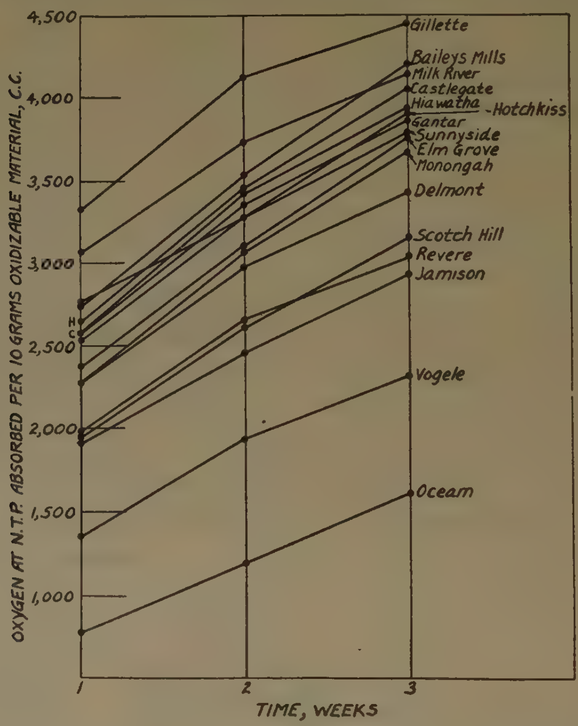


FIG. 1.—ATMOSPHERIC OXIDATION AT 150° C., OXYGEN USED.

the errors due to preoxidation are not so serious, and the reactivity determinations with such coals have a fair degree of accuracy. It is apparent therefore that reactivity toward oxygen, for high-rank coals at any rate, is a true measure of the stage in the "progressive carbonization" at which the ulmin ingredient has arrived, that is to say, it may be used as a measure of the rank of a coal. The most reactive coals are lowest in rank, the least reactive are highest. A graphic illustration of the oxidation of these coals is given in Fig. 1.

Table 3 shows that the percentage rate of production of soluble ulmins places the coals in a similar order and may therefore be used to

determine reactivity. Any oxidation which occurs during the preliminary treatment helps in the production of soluble ulmins and is therefore measured. The determination is not particularly accurate in the early stages of oxidation because there takes place a certain amount of colloidal dispersion in alkaline solution before true solution, due to the formation of carboxylic groupings in the ulmin molecules. The amount of dispersion is a factor difficult to estimate or control, so that little reliance can be placed on figures showing less than 20 per cent. of soluble ulmin. The results are shown graphically in Fig. 2.

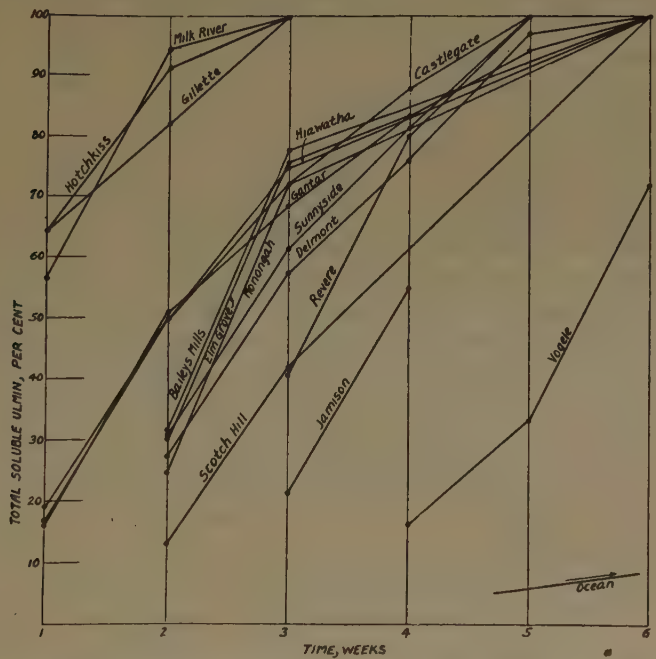


Fig. 2.—Atmospheric oxidation at 150° C., soluble ulmin formation.

In comparing the rates of formation numerically the simplest way is to compare the times required for the production of 50 per cent. solubility. At this point the determinations are reliable, and since the shape of the curve is much the same for each coal, the error introduced by so doing is not great. The number of hours taken for the production of 50 per cent. solubility places the coals in an order such as would be expected from their analyses and measurement of oxygen used. The Hotchkiss coal is placed in a more probable position than was obtained for it in the direct measurement of oxygen used, since in this case the oxidation which took place before the experiment helped in the production of soluble ulmins.

In the grouping by measurement of soluble ulmin formation, there is a well-marked difference between the coals of very low rank, Gillette, Hotchkiss, and Milk River, and the next lowest in rank, the Gantar. The figures given for Baileys Mills, Elm Grove and Monongah coals should not really be included in this table, since these coals were oxidized at a lower temperature before the other coals. To make them at all comparable a small sample of one of them was oxidized at 150° C. and a factor deduced for the calculation of the previous results at 134° C. to the present temperature of 150° C. It appears that the shape of the former curves differed at the lower temperature, being flat for a longer time and then shooting up more quickly than the curves at the higher temperature. The values showing soluble ulmin formation at the end of the second week suggest that there is really little difference between the reactivities of the Gantar, Hiawatha, Castlegate, Baileys Mills and Elm Grove coals.

There is a sharp line of demarcation between the Delmont and Revere coals. Scotch Hill coal ranks next to Revere. Far removed from Scotch Hill comes Jamison, and then Vogele. Least reactive is the Ocean mine coal. Between the highest four members of this series the differences in reactivity are extreme and are sharply defined both by the direct determination by measurement of oxygen used and by the indirect determination by measurement of the formation of soluble ulmins.

TABLE 3.—*Atmospheric Oxidation of Coals at* 150° *C.*

[Percentage Rate of Formation of Soluble Ulmin]

Coal	Time for One-half Ulmin Formation, Hr.	Time, Weeks									
		1	2	3	4	5	6	7	9	11	15
Gillette	140	64	82	100							
Hotchkiss	140	64	91	100							
Milk River	150	56.5	94	100							
Gantar	332	16	51	68.5	83.5	94	100				
Hiawatha	334	16.5	50	74.5	100						
Castlegate	334	19	50	72	88	100					
Baileys Mills	390		31.5	77.5			100				
Elm Grove	410		30	75			100				
Sunnyside	417		30.5	61.5	81	100					
Monongah	425		24.5	72			100				
Delmont	430		27.5	57.5	76	97	100				
Revere	505		22.0	51.0	80	100					
Scotch Hill	550		12.6	42.0			100				
Jamison	650			21.5	55			100			
Vogele	915				16	33	72		100		
Ocean	1850									50	100

Oxidation by Nitric Acid

Oxidation by nitric acid is in many ways the most interesting of the three methods for determining reactivity toward oxidation. The velocity of reaction may be increased almost indefinitely by increasing the concentration of nitric acid used. The concentration of acid used can be measured easily by titration, and the reagent does not deteriorate on standing.

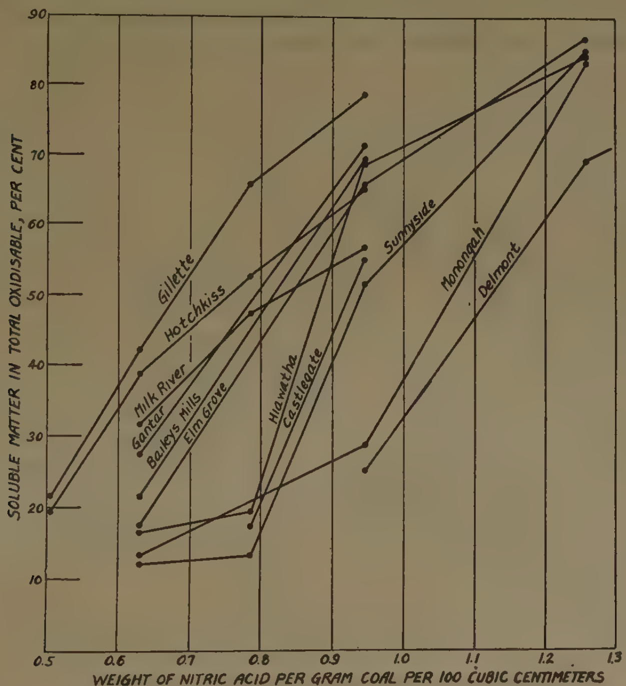


Fig. 3.—Nitric acid oxidation of low-rank coal.

Details of the oxidation with nitric acid are given in Table 4 and illustrated graphically in Figs. 3 and 4. For a moment, leaving the low-rank coals out of consideration, it will be seen that nitric acid is a very discriminating oxidizing agent. This is especially the case in coals ranging from 82 to 90 per cent. in carbon content, the range most useful for commercial purposes.

The general arrangement of the coals from consideration of their reactivity toward nitric acid is again approximately the same as that obtained from considerations of other reactivity determinations. Ultimate analysis is not a reliable or discriminating agent to use, and atmospheric oxidation is a tedious process inundated with difficulties, so that in the use of nitric acid there is a better chance of determining the

reactivity of any coal by a quick laboratory method and of obtaining a rational classification. True it is that the procedure necessitates careful standardizations, otherwise there is a possibility of serious errors creeping in, but in view of the discriminating nature of the reaction it is apparent that the results are more reliable than those obtained from atmospheric oxidation.

During the determination of reactivity by air or nitric acid it is also possible to estimate the relative proportion of plant remains and ulmins in the coal from the weight of residue after treatment and the amounts of ash present in the residue and original coal.

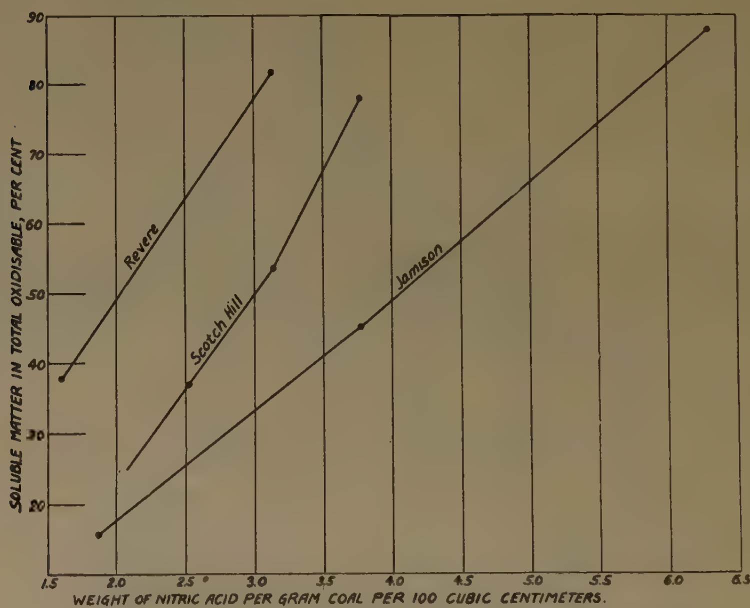


FIG. 4.—NITRIC ACID OXIDATION OF INTERMEDIATE COALS.

In order that the reaction should be completed in a few hours the concentration of nitric acid used was varied—the higher the rank of the coal the stronger the acid used. The upper limit of concentration must not be very great, as the cuticles and spore exines in the residue that are comparatively inert to mild oxidation may be attacked by boiling with a nitric acid solution of high concentration. It was thought undesirable to use solutions of nitric acid stronger than normal. Since some of the coals did not become soluble in alkalies after boiling 8 hr. with normal nitric acid small amounts of potassium chlorate were added. This gave in effect a dilute Schultze's reagent, but it was only necessary for high-rank coals.

TABLE 4.—*Action of Nitric Acid on Coals; Percentage Rate of Ulmin Formation*

Coal	Concentration per 100 c.c. for 40 Per Cent. Ulmin Formation	Concentration in Grams HNO_3 per 100 c.c. Solution per Gram Coal												Normal Acid, g. $KClO_3$	
		0.504	0.63	0.787	0.945	1.26	1.575	1.89	2.52	3.15	3.78	4.5	6.3	0.65 g.	1.3 g.
Gillette	0.620	21.5	42.1	65.6	78.7					100			100		
Hotchkiss	0.640	19.3	38.8	52.5	64.7							100	100		
Milk River	0.715	13.9	31.5	47.5	56.5							100	100		
Gantar	0.730	10.3	27.8		71.7							100	100		
Baileys Mills	0.755		21.8		68.8	84.3		89.9					100		
Elm Grove	0.760		17.7		65.0	86.6		89.4					100		
Hiawatha	0.850		16.4	19.8	69.4								100		
Castlegate	0.880			17.7	54.3								100		
Sunnyside	0.900		12.0	13.0	51.7	85.0							100		
Monongah	1.01		13.3		28.8	83.7							100		
Delmont	1.05				24.6	69.5	73.8						97	100	
Revere	1.78						38.0			81.6				100	
Scotch Hill	2.53								37.0	52.9	78.0			100	
Jamison	3.45							15.6			44.6		88		100

No Ulmin up to Normal Acid	100 c.c. N/2 HNO_3 + N g. $KClO_3$				100 c.c. N HNO_3 + M g. $KClO_3$			
	N = 1.3 g.	N = 2.6 g.	N = 5.2 g.	N = 6.5 g.	M = 1.3 g.	M = 2.6 g.	M = 5.2 g.	M = 6.5 g.
Vogele 6.3	24	84.3		98	100			
Ocean 6.3			43.0	64.3		85		100

The time factor is the only limit to the dilution which may be used. A period of 8 hr. was thought most suitable for the reaction. This is sufficiently large to negative any discrepancies due to such factors as difference in the ease of wetting the coal particles and is sufficiently short to allow the test to be completed in the average working day. Since one object of this work is to evolve a system of examination of coal suitable for the ordinary routine laboratory, the desirability of limiting the time to 8 hr. will be appreciated.

The safest way of examining coals for reactivity is to make a number of determinations of soluble ulmin formation with different concentrations of acid and to plot a graph of the results, plotting percentage of soluble ulmin formation against concentration of acid. The important points are the central portions of the curve and the end points. For routine purposes it will be sufficient to determine a few points near the center of the curve and the final point. The analysis of the coal under consideration and the results given in Table 4 will usually enable one to do this with little previous experience in the methods by three or four snap experiments.

In order to obtain the end point it will be sufficiently accurate to determine the weight of residue obtained by extraction with alkalies after boiling for 8 hr. with about five times the concentration of acid necessary to convert about 50 per cent. of the oxidizable material into soluble matter. If the coal is a normal coal of low or medium bituminous rank, 100 c.c. of normal acid per gram will usually be sufficient. Should the coal be of higher rank than this and only go partly into solution with normal acid it will be necessary to add small quantities of potassium chlorate to determine the end point. The more difficult it is to make the coal soluble the more chlorate should be added. The highest member of this class of coals will probably be oxidized sufficiently by boiling with 100 c.c. of normal nitric acid and 2.5 g. in potassium chlorate per gram of coal for 8 hours.

With higher rank coals, that is, those that will give no alkali-soluble ulmins with normal nitric acid under these experimental conditions, it will be necessary to plot a curve using throughout a mixture of nitric acid and potassium chlorate. Two such coals, Vogele and Ocean, are shown in Table 4. Experience will be the guide in deciding the concentration to use for these high-rank coals, and a graph is the safest way of determining the end point.

During the present experiments the weight of residue obtained during atmospheric oxidation was available as a check. The figures given in Table 5 will show that the "snap" determinations which were made in the majority of instances give values for the proportions of residues which are of the same order as those obtained by atmospheric oxidation. These values, strictly speaking, are not directly comparable,

since ordinarily the same sample of coal was not used in the nitric acid oxidation as in the corresponding atmospheric oxidation. Two samples were taken from the same lump of coal and treated separately, the one sample for atmospheric oxidation, the other for nitric acid oxidation. Such variations as are shown in Table 5 for percentages of plant remains in each corresponding pair of samples are of the order to be expected in contiguous portions of the same lump of coal.

TABLE 5.—*Rational Analysis of Coals by Solvents and Oxidation*

[Percentages on Ash-free, Dry Basis]

Coal	Resins and Hydrocarbons, Per Cent.	Air Oxidation		Nitric Acid Oxidation		
		Ulmins (Oxidizable Material), Per Cent.	Plant Remains (Inerts), Per Cent.	Ulmins (Oxidizable Material), Per Cent.	Plant Remains (Inerts), Per Cent.	
Gillette	3.8	91.1	5.1	92.2	4.0	at 70
Milk River	3.8	92.4	3.8	93.2	3.0	
Hotchkiss	3.4	95.4	1.3	95.7	1.0	c.c.
Gantar	4.0	86.8	9.2	88.7	7.3	per g.
Baileys Mills	5.4	85.9	8.7	86.8	7.8	
Elm Grove	6.4	86.0	7.6	85.2	8.4	
Hiawatha	5.7	87.3	7.0	87.6	6.7	
Castlegate	5.8	82.6	11.6	80.8	13.4	
Sunnyside	5.7	90.3	4.0	89.6	4.7	
Monongah	3.9	87.5	8.6	87.0	9.1	
Delmont	3.9	88.1	8.0	85.2	10.9	
Revere	3.5	87.3	9.2	90.4	6.1	
Scotch Hill	4.4	86.9	8.7	84.6	11.0	
Jamison	3.2	88.0	8.8	88.4	8.4	
Vogele	1.6	95.7	2.7	95.4	3.0	
Ocean	<1.0	89.5	9.5	90.5	8.5	

NATURE OF ACTION OF NITRIC ACID

The nature of the oxidizing action of nitric acid must now be considered briefly because only by doing so can one explain the behavior of the different coals under treatment. Nitric acid may react in three ways during this action: it may oxidize, nitrate, or do both. Organic chemistry indicates that usually nitration does not take place unless concentrated acid is used, a dehydrating agent is present, or certain groupings occur in the reacting molecule which facilitate the entrance of the nitro-grouping. During the present experiments the dilution of the reacting nitric acid is sufficient to preclude the formation of nitro-groupings in the ulmin molecule unless the molecular arrangement is such as to facilitate this type of reaction. This must be so because

nitration undoubtedly does take place, as well as oxidation. Three pieces of evidence support this statement.

In the first place, there is at first an increase in the weight of the coal during the action of nitric acid. During the atmospheric oxidation of low-rank coals, such as Gillette or Hotchkiss, there is no increase in weight, and with high-rank coals, which normally show an increase in weight during the initial stages of atmospheric oxidation, the increase in weight with nitric acid is much greater than with air.

Secondly, titration of the reacting acid before and after oxidation has shown that the solubility produced in the ulmin is much greater per unit weight of "available" oxygen in the oxidizing agent than with the same volume of oxygen used in straight oxidation experiments.

Thirdly, after the formation of soluble ulmins by the dilute acid used in these experiments there is a general breakdown of the ulmin molecule into water-soluble products, which are mixtures of acids and nitrophenols.

The enhanced solubility of the ulmin molecule coincident with the entry of nitro-groupings into the molecule may be due to the formation of pseudo-acids, a fact which is confirmed by the slow rate of solution of these compounds in alkaline solutions. It may also be due to the presence of phenolic-like complexes, which are soluble in alkalies when nitro-groupings are present in suitable positions.

The total solubility of the ulmin oxidized by nitric acid depends on the sum of the solubilities due to the presence of (*a*) groupings which may act as pseudo-acids; (*b*) nitro-groupings situated in the correct juxtaposition to phenolic hydroxyl groupings; (*c*) carboxylic groupings due to straight oxidation. Of these three, straight oxidation always tends to the production of carboxylic groupings and therefore oxidation alone always increases the solubility of the molecule. The presence of nitro-groupings, however, may not always help to increase the solubility, since in this case solubility depends on the presence of other groupings such as oximes or hydroxyl in the correct position. Should these groupings be absent or be changed during oxidation, increasing nitration may not cause increasing solubility and may even tend to reduce the solubility of the molecule.

Apparently this happens with certain low-rank coals, the Hotchkiss and Milk River coals being typical of this phenomenon. With low concentration of nitric acid—that is, when straight oxidation is certainly the principal effect of the nitric acid—these coals, being more reactive than the higher rank coals such as Baileys Mills and Elm Grove, become more soluble in alkaline solutions. With higher concentrations the nitrating action of nitric acid becomes more prominent, and nitro-groupings are introduced into the molecule. Suitably placed groupings are not present to give enhanced solubility to the molecules of the low-rank ulmins, hence the rate of formation of soluble products falls off, even though the

reaction with nitric acid may still be proceeding more quickly than with the higher rank coals.

The point at which the formation of alkali-soluble ulmins in low-rank coals becomes no longer a reliable guide for the rate of the reaction under the conditions of these experiments is when about 30 or 40 per cent. of the maximum solubility is reached. At 40 per cent. of solubility it is possible to determine with fair accuracy the amount of soluble ulmin present, so that in order to obtain a simple numerical value to express the reactivity of each coal towards nitric acid it will suffice to determine from the graphs the concentration of nitric acid which will produce 40 per cent. of alkali-soluble ulmin. This has been done, and the values for each coal are shown in the first column of Table 4.

These numbers illustrate the increasing inertness of the ulmins as their ranks increase. In order to avoid the possibility of obtaining misleading results due to nitration, particularly with low-rank coals, a new method is being developed using mixtures of dilute hydrochloric acid and potassium chlorate as oxidizing agents. The extracted coals are heated with this reagent in pressure bottles by means of boiling water and the soluble ulmins determined in the usual way. The results so far obtained indicate that results of the same type are obtained as with nitric acid and atmospheric oxidation, but for low-rank coals they are much more accurate and discriminating.

Summary

As a result of the action of oxidizing agents and solvents on coals we can therefore measure both types of differences which exist between them. Organic solvents, pyridine, chloroform and pentane for example, enable estimations of the percentages of resins and hydrocarbons to be made. For the purpose of this paper no attempt has been made to separate the resins from the hydrocarbons by means of pentane. The results of the action of oxidizing agents on the extracted coals are twofold. From the weight of residue, insoluble in alkalies after prolonged oxidation, the relative proportions of inerts, chiefly cuticles and spore exines, and reactive constituents, almost entirely ulmins, may be determined. During the reaction an estimate may be made of the order of reactivity of the ulmin constituents.

The estimation of the relative proportions in which the ingredients exist is the first step in a scientific classification of coal and is not possible by any other methods than are mentioned in this paper. Measurement of the reactivity of the ulmins is the second step, grouping the coals in an order similar to that obtained by other classifications, which depend on other properties of the coals. The decreasing reactivity of the ulmin with increasing rank is a direct result and measure of the progressive carbonization which occurs with this major constituent of coals. It is

therefore a true basis for classification as defined by John Stuart Mill, and moreover its determination is independent of the relative proportions of other ingredients that are present. In this respect it has the advantage over other systems of classification which can only measure the total properties of groups of mixtures whose relative proportions vary and whose properties are not additive. Table 6 compares the reactivity measured by the action of nitric acid with rank measured by two of the older methods.

The relationship between reactivity and the other properties of the ulmins, as also with the relationship between any one other property and the rest, may strictly hold with coals of the same age. It is impossible to discuss fully this phase of classification in the space available here, but one should always look askance at a comparison between coals of different geological periods by any one property until confirmed by an examination of other properties. It is unlikely that very great discrepancies will be encountered in normal coals. A "normal coal" may be defined as one formed by the accumulation and decay of forest growth, first of all passing through a "peaty fermentation" to break down the woody elements, then having sufficient time to make possible the complete combination of these degradation products to form peat ulmins before being subjected to the short-duration high temperatures or long-continued moderate temperatures which cause the changes of condensation, polymerization or both that are known as "progressive carbonization" and determine the "rank" of the coal. David White[3] has suggested that temperature and time may be interchanged. This is no doubt approximately true but it is quite apparent that coals which have been subjected to sudden changes in temperature before the complete transformation of the woody structures into ulmins may behave abnormally. It is impossible to predict abnormality from our present geological knowledge so that no one system of classification can be relied on to give satisfaction for all ages of coals.

The rational analysis and determination of reactivity may be relied on to classify and compare coals of the same age in a discriminating manner. It can not yet be considered to have been proved reliable for coals of widely different ages because of the possibility of the occurrence of abnormalities. The safest way is to examine coals in two different ways, first of all to determine rank approximately by, for instance, ultimate analysis, then to determine the reactivity of the ulmins and the relative proportions of ingredients present in the coals by the regulated action of nitric acid. The results will check for all normal coals. Those which do not check may be considered abnormal and will be expected to have different properties.

---

[3] David White: Progressive Regional Carbonization of Coals. *Trans.* A. I. M. E. (1925) **71**, 253.

TABLE 6.—*Correlation of Nitric Acid and Other Systems of Classifying Coals*

Geological Formation	U. S. Geological System Fuel Ratio		Nitric Acid Oxidation		Seyler's Classification	
	Name	Fixed Carbon Volatile Matter, Per Cent.	Coals	Concentration per 100 c.c. for 40 Per Cent. Solubility in 8 Hr.	Limits of C and H	Name
Fort Union	Subbituminous	1.07	Gillette	0.620	C, < 75 per cent.	Lower than Lignitous
Lance		1.21	Hotchkiss	0.640	C, < 75 per cent.	Lower than Lignitous
Fort Union		1.34	Milk River	0.715	C, < 75 per cent.	Lower than Lignitous
Fort Union		1.61	Gantar	0.730	C, 75 to 80 per cent.; H, 5 to 5.8 per cent.	Ortholignitous
Monongahela	Low-rank bituminous	1.05	Baileys Mills	0.755	C, 75 to 80 per cent.; H, 5.8 per cent.	Perlignitous
Monongahela		1.26	Elm Grove	0.760	C, 80 to 84 per cent.; H, 5 to 5.8 per cent.	Metalignitous
Mesa Verde		1.17	Hiawatha	0.850	C, 80 to 84 per cent.; H, 5 to 5.8 per cent.	Metalignitous
Mesa Verde		1.37	Castlegate	0.880	C, 80 to 84 per cent.; H, 5 to 5.8 per cent.	Metalignitous
Mesa Verde		1.41	Sunnyside	0.900	C, 80 to 84 per cent.; H, 5 to 5.8 per cent.	Metalignitous
Monongahela	Medium-rank bituminous	1.51	Monongah	1.01	C, 80 to 84 per cent.; H, 5 to 5.8 per cent.	Metalignitous
Monongahela		1.45	Delmont	1.05	C, 84 to 87 per cent.; H, 5.8 per cent.	Perparabituminous
Monongahela		2.08	Revere	1.78	C, 84 to 87 per cent.; H, 5 to 5.8 per cent.	Parabituminous
Monongahela		1.82	Scotch Hill	2.53	C, 84 to 87 per cent.; H, 5 to 5.8 per cent.	Parabituminous
Monongahela		1.89	Jamison	3.45	C, 84 to 87 per cent.; H, 5 to 5.8 per cent.	Parabituminous
Monongahela	High-rank bituminous	2.58	Vogele	>6.3	C, 87 to 89 per cent.; H, 5 to 5.8 per cent.	Orthobituminous
Monongahela	Semibituminous	4.34	Ocean	>6.3	C, 89 to 91.2 per cent.; H, 4.5 to 5.8 per cent.	Submetabituminous

An intelligent use of a classification whose basis is reactivity therefore conforms with the ideal of Huxley in that it is the "actual or ideal arrangement together of those members which are like and the separation of those which are unlike." So far even in this early work the methods are apparently giving results of the right nature. With increasing experience the methods may be modified and improved so that we may at last hope to have a rational classification of coal.

Acknowledgment

Thanks are due to assistance rendered in this work by Harold M. Morris, Research Fellow, Carnegie Institute of Technology.

DISCUSSION

R. Thiessen, Pittsburgh, Pa.—Dr. Francis says that the resins are not extracted with the alkalies after the oxidizing reagents; I have found that they disappear.

W. Francis.—In presenting my paper, I was unable to give full details of the conditions under which the oxidations took place. I am careful to remove the resins and hydrocarbons before I oxidize the coals. I realize these substances might tend to give misleading results and as I am intent on studying only the relationship between reactivity and rank of the coal ulmins I try to eliminate as far as possible as many factors irrelevant to the main issue.

With regard to the action of nitric acid on coal resins, there is certainly some action with the formation of resinous acids of unknown composition but which certainly are soluble in alkalies.

With regard to the oxidizability of coal resin in air, it has been quite definitely shown by my experiments that a resin does not oxidize so quickly as its containing coal at low temperatures. At higher temperatures it may oxidize more quickly.

W. T. Thom, Jr., Princeton, N. J.—Dr. Francis gave the percentage of the carbon in the various coals. I should like to ask him the precise form of analysis from which those percentages were obtained and also to what extent he discriminates or thinks a discrimination might be made between the carbon which is ordinarily reported as fixed carbon and that which is included in constituents ordinarily classed as volatile matter. It seems to me that the oxidizability or stability of the carbon may well depend on surface effects, and that different surfaces for action may be exposed by carbon in the groundmass or what one might regard as more solid portions of the coal, as compared with carbon of the volatile constituents.

A difference in exposed surface is, I believe, to be invoked as an explanation for the difference in behavior of the very low-rank coals and the higher rank coals. The higher rank coals, having been compressed more, are less porous and therefore offer smaller surfaces to oxidation or similar surface reactions.

W. T. Thom, Jr.—What I should like to obtain is your opinion as to the relation between speed of reaction and the surface condition of the reacting carbon. Are your percentages based on air-dry analysis or upon ultimate analysis?

W. Francis.—The coals were analyzed in the regular Bureau of Mines manner. The procedure, I believe, is to take the sample of coal, grind the whole of it, then to make the analyses in the air-dried state and calculate from these and the percentages

of ash and moisture found, the ash-free dry analyses. The carbon figures I gave were obtained in this way.

W. T. Thom, Jr.—They were calculated to a dry basis?

W. Francis.—Yes. As regards your point, I am not quite clear whether your question referred to the influence of the physical state upon the oxidation or upon destructive distillation.

W. T. Thom, Jr.—Toward oxidation.

W. Francis.—Yes, the physical nature of the surface is of very great importance with regard to oxidation and that is another reason why, before oxidizing the coals, I separate them from the resins and hydrocarbons. In order to do this the coals are extracted with pyridine and chloroform. Normally speaking, coal is not susceptible to the attacks of a pure solvent like chloroform, but after the action of pyridine, which has some ill-defined effect on the colloidal material and apparently serves to open the pores of the coal, this solvent will dissolve out the resins and hydrocarbons which are present.

I think it is fairer to treat coals in this way because if the differences in surface state are largely of colloidal nature, and if the colloidal aggregates are resolved in the same way, the surface state of the particles should be more uniform in nature.

W. T. Thom, Jr.—The purpose of my questions has been to bring out the relation between the chemical reactivity of different coals and their normal moisture content. The range of carbon percentages in Dr. Francis' tables was, I believe, from 72 per cent. for Gillette subbituminous coal to 89 per cent. for Pittsburgh low volatile coal, and as I recall it, the oxygen absorption was six times greater for the Gillette coal than for the Pittsburgh. If we include moisture, however, as an ingredient of coals, we would find the relative percentages of carbon in the two coals as about 55 to 87, instead of as 72 to 89, and the two curves of carbon content and reactivity would, I believe, run much more nearly parallel. That is a matter of practical importance which appears to emphasize the significance of the moisture content of coals.

H. J. Rose, Pittsburgh, Pa.—One of the most interesting points to me is the excellent agreement between the degree of oxidizability and the fixed carbon content. The coals line up in a fairly orderly way with but few exceptions. I wonder if that might not be due to the fact that Dr. Francis used a close approach to pure coal. As I understand it, the coals used for these experiments were selected anthraxylon.

W. Francis.—They were unpicked seam samples of coal.

H. J. Rose.—Extracted to remove resins and hydrocarbons?

W. Francis.—That is all—then sieved and dried. I do not think that point is a valid one because the coals ranged in ash content from about 1 up to 14 per cent. in some cases. I believe the reason the figures agree so well is that they are all calculated per unit weight of oxidizable material. We are able to do this because during the experiments estimates were made of the proportion of ulmins present in each coal.

H. J. Rose.—Not on the basis of total coal?

W. Francis.—Not on total coal at all because the proportions of different ingredients present varied greatly. These coals contained ulmins varying in percentage from about 80 up to 96. Suppose one of 80 per cent. ulmin content absorbed 2000 c.c. of oxygen and one of 96 per cent. ulmin content also absorbed 2000 c.c. of oxygen;

it is not fair to say that those ulmins are reactive to the same extent. For comparative purposes the volumes of reacting oxygen must be divided by the proportions of ulmins present. The higher the rank of coal, the better the relationship between reactivity and rank holds, because the differences due to the physical state of the particles are then less important. The most difficult ones to treat are the low-rank coals from the West.

Commercial Classification of Coal*

By F. R. Wadleigh,† New York, N. Y.

(New York Meeting, February, 1928)

It is generally realized and very often admitted by both producers and consumers of coal that there is great need for a revision of existing commercial classifications, and this will involve, of course, determination of a standard classification and a constant revision of marketing practices as they are affected by coal classification.

I recently sent out some 200 letters to producers and consumers with a request for their views on the subject of commercial classification. Nearly all of the replies from both sides were that there was needed some change in the present method. There were three or four dissenting voices but the great majority of opinion was in favor of the formulation of some standard commercial classification.

The classifications in commercial use today can best be described as a hodgepodge. They are made upon different bases, have different names and have different ways. There are, for instance, about 11 single bases of classifications as used commercially and some 17 or 18 combination bases. Then there were the Tidewater Pool Exchanges. The classification which they put in force is still used in the tidewater markets and has several different bases of its own.

It will be considerable of an undertaking to bring about a change in marketing methods as affected by coal classification but it is well worth doing and I think the committee which has been appointed under the auspices of the American Society for Testing Materials realizes the importance of the subject and is on the way to at least bring about some provision for adjusting matters. By way of starting, each member of the Committee on Commercial Classification has been assigned to a general market district and each member is now engaged in collecting the commercial market practice data for submission at the next meeting of the Committee.

About a year ago I made up a list of the commercial classifications in the different markets and the total was 171. The 11 single bases I found used in commercial practice were:

* The author's statements are based on a report on Commercial Classification of Coal which was prepared for the American Society for Testing Materials and was also submitted to the A. I. M. E. Coal and Coal Product Committee.

† Consulting Engineer.

1. Geological.
2. Chemical.
3. Combustion characteristics.
4. Carbonization characteristics.
5. Physical appearance.
6. Geographical location.
7. Size.
8. Company handling; company names.
9. Trade names.
10. Uses.
11. Use methods.

The 16 combination bases were:

1. Geographical and geological.
2. Geographical and chemical.
3. Combustion characteristics and carbon content.
4. Carbonization characteristics and analysis.
5. Physical and carbonization characteristics.
6. Size and use.
7. Company name and seam.
8. Size and analysis.
9. Size and size of screen.
10. Chemical and use.
11. Geographical and use.
12. District and use.
13. Size and use.
14. Geographical and size.
15. Seam and size.
16. Trade name and use.

Classification of Coals from the Point of View of the Railroads

By Eugene McAuliffe,* Omaha, Nebr., and Malcolm Macfarland,† New York, N. Y.

(New York Meeting, February, 1928)

Our North American railway system, including the lines serving the United States, Canada and Mexico, with a total operating mileage of 303,040, employing 71,818 locomotives, represents not only the greatest industrial user of fuel coal, but in the variety of coals used, ranging from anthracite down through the several gradations of bituminous coals, as well as lignites, covers about all there is in American-mined coals.

Necessarily the wide range of fuel coal embraces not only extremes of calorific value but also every possible gradation shown in size and preparation, from run of mine to washed nut and slack. Approximately 8 per cent. of the fuel used by the railroads is consumed in shop power plants, shop buildings, pumping stations, station buildings, etc. The principal class of coal (as designated) used by the railroads is bituminous, such, however, including a substantial tonnage of subbituminous and lignite coal; all United States railroads using in 1923 for locomotive, shop, station buildings and miscellaneous uses, a total of 4,578,000 tons of anthracite, 155,795,000 tons of bituminous, subbituminous and lignite coal and 58,005,000 bbl. (42 gal.) of fuel oil.

The Class 1 railroads of the United States (those whose gross annual earnings equal or exceed $1,000,000) used in 1926:

Bituminous, subbituminous and lignite coal, net tons.....	140,083,885
Anthracite, net tons..................................	3,667,505
Fuel oil, gal...	3,058,915,511

Railroad fuel-purchasing agents rarely attempt to classify coal for locomotive use on the basis of chemical analysis; although this is very generally used as a guide in the initial selection of a fuel supply. The extraordinarily high rate of combustion developed in the modern locomotive, generating as high as 2500 b.hp., with limited facilities for cleaning fires en route, suggests the importance of a fuel supply containing a reasonably low percentage of ash-making material, and while the percentage of locomotive coal handled through automatic stokers is growing, the principal tonnage is yet fired by hand.

* Special Representative, Union Pacific System.

† General Fuel Inspector, New York Central Lines.

Many railway fuel-purchasing agents divide the bituminous coals into three primary classifications: low volatile, medium volatile and high volatile. When antismoke regulations are enforced, low volatile, anthracite or coke must be provided. As these three grades of fuel lack the flexibility common to medium or high-volatile coal, special draft arrangements must be provided for the locomotives, and in addition an extra measure of supervision over locomotives and crews is generally provided.

This rather general division is merely preliminary to a closer selective classification quite generally employed in determining the fuel value of coal received from competing mines or districts. This second and more scrutinizing check is predicated on the factors of: (1) physical structure; (2) method of preparation and mining; (3) thermal content expressed through proximate analysis.

System for Indicating Value of Fuel Coals

The coal coming from each mine, as well as the method of mining and the facilities for cleaning and preparing it, are carefully examined and face or carlot samples are analyzed for moisture, ash, fixed carbon, volatile combustion and sulfur content, and the B.t.u. value of the coal as received is determined. An analysis of carlot samples taken uniformly, approximating the track-scale weighing period, offer the best relation between thermal value per pound and the volume of fuel paid for by the purchaser. The following system for indicating the relative physical and analytical value of railroad fuel coals is submitted for consideration:

A. Best coal physically—firm structure.
B. Second quality physically—mining and preparation requires care.
C. Friable structure—fines tend to excessive proportion.
D. Mining methods and preparation not satisfactory.
E. Natural conditions prevent satisfactory preparation.
1. Best coal chemically—low ash, low sulfur.
2. Good coal chemically—preparation requires care.
3. Fair coal chemically—preparation requires special care.
4. Not satisfactory chemically.
5. Unsuitable chemically.

Each coal is given a combination letter and numeral designation; *e. g.*, *A*-1, *B*-3, etc.

Grading for Various Branches of Service

Those in charge of railway fuel coal find it necessary and desirable to segregate and thereafter assign certain grades of coal to the various consuming branches of the service; as, for example, to:

Main line passenger, express and mail trains.
Divisional and branch passenger, express and mail trains.
Freight trains.
Switching locomotives.
Locomotives operating within anti-smoke regulation zones.
Power plants.
Station buildings, water-pumping plants, etc.
Flag and switch shanties, cabooses, etc.
For blacksmithing use.

The purchase of fuel by a railroad company presents a problem quite different from that of the ordinary commercial consumer, who is governed in the majority of cases by the relative B.t.u. that can be obtained for each cent expended. In the case of a railroad, the cost varies widely. When the coal is bought from mines located on the rails of another company, the published rate must be paid in cash to the foreign line or lines handling it. On the other hand, when the coal is mined on the user's rails, the cost of transportation from mine to point of consumption is invariably less, representing what is known as the "out-of-pocket cost." Again, the purchase by a railroad of its fuel coal from mines on its own rails assists in the general prosperity of the mines and mining communities supplying it. These, with other conditions, make frequent compromises necessary; that is to say, it may be cheaper to buy a somewhat inferior coal from mines served by the purchasing company, rather than to pay for a better quality obtainable elsewhere.

Consumption of Inferior Grades

To meet a situation such as this (which is not infrequent) locomotives and power plants are designed to consume the inferior grade of fuel, the result being a lower ultimate cost to the railroad. Such conditions, plus continuity of delivery, an available empty coal-car supply, a return loading for empty cars, and the governing direction of heaviest tonnage and empty car movement, all represent factors that must be put in the scale and carefully weighed together with the percentage of volatile, moisture, ash and thermal value and other characteristics shown for different coals. The final determination is often made by actual road tests, the relative value of the fuels tested determined by the pounds used per 1000 gross tons moved one mile, as well as water evaporated from and at 212° F. Here again, unavoidable variations enter, including stand-by losses, coal used in firing up and after arrival at terminals, the effect of wind resistance, length of train and character of lading, all of which, while not precisely measurable, must be considered and allowed for by those in charge of tests.

In Texas and in certain northwestern states large bodies of lignite coal, cheaply available, are drawn upon by many railroads. Necessarily, these lignites, carrying a very high moisture content and responsible for heavy spark losses, show low gross ton-mile movement results, quite comparable with their low thermal value. Here again the question of road haul enters: the deposits of lignites used by the railroads being principally valuable on account of their proximity to points of consumption; better fuels being available only at the expense of excessively long hauls.

It may be said that the determination of a fuel supply by a given railroad depends on many conflicting factors, certain of which preclude the observance of refined classification regulations. Where such can be used, they are in general effect, more particularly by the roads fueled from mines located in the great Appalachian coal-bearing region.

DISCUSSION

H. J. Rose, Pittsburgh, Pa.—Is the classification given in actual use or is it only tentative?

E. McAuliffe.—It is tentative and is used with variations by certain railroads but it is not at all general. I have no hesitation in saying that the commercial elements that enter into the purchase of coal in the majority of instances transcend their thermal characteristics and values.

W. H. Blauvelt, New York, N. Y.—Mr. McAuliffe says that the commercial characteristics and conditions rule. Of course, they rule in everything, in the final analysis. Are the railroads doing anything in the way of research to show that coal that costs $2 is sometimes cheaper than the coal that costs $1.50? In other words, are they only buying from the purchasing agent's point of view, or are they buying with the help of their research department? Some day, are we not going to buy coal on the basis of what the scientists, the chemists and the research men and the geologists tell the purchasing agent about the coal he is considering? What is the likelihood of the railroad's getting to that point of view?

E. McAuliffe.—From my experience in the purchasing and handling of railway fuel, I doubt that chemical characteristics will ever take a leading place in the purchase of railroad fuel coal, for the reason that the questions of location, transportation, continuity of supply, etc., carry wider variations than we normally find in thermal values. For example, when a railroad finds it advantageous to use an inferior coal, its engineers study and weigh the situation and forthwith proceed to adapt the design of locomotive to meet the requirements of the fuel. As a specific case, the Northern Pacific Railroad obtained its Western territory fuel supply from its mines in the State of Washington and in the Red Lodge, Montana, district for a great many years. These two fields were becoming more expensive, more costly to produce. The railroad knew it had a tremendous acreage of low-grade lignite in the Rosebud District of Montana, so it made a study of that situation, mined a sufficient quantity, hauled it to the railroad, a distance of perhaps 35 miles, and ran some tests. Then the mechanical engineering department having developed the most modern locomotive, attempted certain further modifications of design, for example, a new design of grate, which made it possible to use successfully in heavy main-line traffic, coal running as low as 7500 B.t.u. per pound.

To further meet that situation the Northern Pacific Railroad recently ordered a locomotive for experimental purposes that will weigh 521 tons when fully equipped for service, and which has a firebox area of 185 sq. ft. So, answering your question again and perhaps more concretely, those certain practical conditions that we have to contend with are of more importance and frequently more easily surmountable, or when surmounted they at least transcend in importance, the variation in thermal content between coals. In substance, if we have to, we can on a railroad burn anything. That has been fairly well demonstrated in the last five or six years.

A. C. Fieldner, Washington, D. C.—In the table of properties of coal that are considered in the purchase of coal for railway fuel purposes, it would seem that one very important property is not considered; namely, the coking or caking properties of the coal as influencing the softening and caking together of coal on the grate. There is a considerable difference in various coals with reference to this property. It has an important bearing on the type of stokers and on the manner of firing. Is that point taken into consideration in the purchase of railway fuel?

E. McAuliffe.—Again change in locomotive design and construction have substantially eliminated that situation. In my younger days, when I was in railway locomotive service, using a small grate area and with locomotives carrying fires from 12 to 16 in. in thickness, the caking qualities of coal were of material importance. Many locomotives were equipped with bars and we were compelled to break the surface crust at frequent intervals in order to effect a sufficiently rapid rate of combustion and get the necessary steam. We have now gone so far from the 16-in. fire that on the Northern Pacific Railroad and on the Sante Fé Railroad, with the so-called round-hole grate, our locomotives are burning coal in what might be called a manner almost equivalent to the pulverized coal process used in central power plants and for burning lime and cement. There again we have found out something in the last few years.

In talking to railroad men and in instructing locomotive men, in past years I, with others, insisted that a locomotive could not be operated successfully and with an average measure of fuel economy with less than a 35 per cent. air opening in the grates. Today the Northern Pacific is using experimental grates with a 7 per cent. air opening. We then thought we had to have a 35 per cent. opening to get the necessary air to consume the coal burned in our small fireboxes. The fact was that we had to put excess coal in in order to clog these very large openings and to keep the fire from turning over under the effect of the strong exhaust blast then used.

We find today that with ½-in. round grate openings we can carry a very thin fire, and as the air filters through the small openings in small jets, there is no turning over effect and the result is that when a locomotive pulling 100 freight cars heads into a siding and the engineer shuts off the steam, we find there is not over 2 in. of fire on the grates. The fire is perhaps 15 or 16 in. deep while the engine is in heavy service, but it is in a very loose condition; in other words, there is about 16 in. of coal floating above the grates by reason of the ingress of these many small streams of air that are uniformly distributed over the firebox area. So, the caking problem has disappeared in our railways.

A. C. Fieldner.—Has ash disappeared as a factor?

E. McAuliffe.—The ash has in a sense dissappeared. On the Union Pacific Railroad, our 7000 passenger locomotives, employed westbound on a continuously ascending run, against the heavy prairie winds experienced in Nebraska and Wyoming, run 556 miles without touching the fire, and at the end of that time the measure of refuse in the firebox is substantially negligible. Why? Because the complete combustion effect reduces and so thoroughly incinerates the noncombustible material that it is largely carried out through the flues. We have a considerable measure of ash

along the railway—so much so that some time ago some of our oil-burning competitors criticized the amount of dust that the trains had to pass through in deep cuts. Again our people met the situation by oiling the sides of the earth cuts and that complaint disappeared.

G. S. RICE, Washington, D. C.—When I was in Iowa, we had coal that made a great deal of trouble in the locomotives. They put limestone on the grates in order to dispose of the ash, so that it was practically slag. Is that a problem today?

E. McAULIFFE.—That is no longer a problem. In my early days we at times shoveled about 500 lb. of crushed limestone on the first fire and that lime, through some chemical mixture under high temperature with the iron, gave us a nonfusing breakable clinker. That is about all there was to the limestone process. Instead of the stuff fusing and filling up the grates and running into the ash pan, it made clinkers that were easily broken. That problem is entirely eliminated today.

G. S. RICE.—Perhaps you do not burn any more Iowa coal.

E. McAULIFFE.—We do not, but there are roads that do. Many of our earlier locomotive troubles were due to small grate areas, with an excessive rate of fuel consumption per square foot of grate, limited heating surface and low steam pressures, in substance an extremely low factor of efficiency.

Use Classification of Coal as Applied to the Gas Industry

By W. H. Fulweiler,* Philadelphia, Pa.

(New York Meeting, February, 1928)

The writer would define the term "Use Classification" as a discussion of the qualities that coal should possess to fulfill the requirements of the industry or process in which it is to be used. The general problem of the gas industry is to transform the maximum amount of the thermal energy in coal into a gas of specified quality.

The qualifications that a coal should possess to be useful in the gas industry are complicated by probably a greater number of factors than in most industries. The most important of these is the delivered price of the coal.

While the general problem of the industry is as given above, yet the actual problem is the production of gas of a specified quality at the minimum cost. The relative cost of coal and the selling prices for the residuals which are influenced to a large extent by geographical locations are of determining importance as effecting the cost of the gas.

It is unfortunate that we cannot disassociate cost figures from a discussion of the desirable characteristics and this results frequently in the use of coal in one location that will be looked upon as quite impossible in another location more favorably situated.

Without going into any discussion of the freight rate problem we might merely point out that in general the freight on the coal is equal to or greater than the mine price. It must be evident, therefore, that price must always be considered in any discussion of the effect of the characteristics of a given coal.

Two Classes of Processes

In discussing the characteristics of coals for use in the industry, we must divide the processes in use into two classes: (1) where the coal is destructively distilled at high temperatures resulting in the formation of coke in the liberation of gas and by-products, and (2) where the coal is consumed by combustion in air or steam with a production of gas and by-products leaving only clinker as a residue. The Germans differentiate these two types of processes in their words "entgasung" and "vergasung."

In the distillation processes, we have the effect of the coal on the yield of gas and its quality; the quantity and quality of coke; and the

* Chemical Engineer, The United Gas Improvement Co.

yield of by-products. Where gas is to be used as city gas we must also take into consideration the effect of the impurities, principally sulfur.

The writer cannot at this time go into a discussion of the effect of the physical characteristics of coal on the character of the coke produced, but may point out that except where the coke is to be used for metallurgical purposes, the important requirements are that it shall not contain an excessive amount of ash and the fusing point of this ash shall not be below a certain minimum.

Factors in Manufacture of City Gas

The more important factor in the manufacture of city gas is the yield and quality of the gas produced. In general, we wish to secure the maximum number of B.t.u. in the gas, and abroad, particularly in Germany, this is used as a single criterion, convenient laboratory methods being now available to determine quite easily the number of B.t.u. feet produced per pound of coal, that is, the number of cubic feet of gas per pound multiplied by the heating value of this gas taken under standard conditions.

With the present general acceptance of the heating value standard, the actual heating value of the gas produced does not become so important a factor as where relatively high values were required when the illuminating standard was in use and it appears that practically any coal of bituminous rank would yield a gas having a heating value greater than 535 B.t.u., which is a figure that appears to be becoming generally accepted as a standard. In looking to our requirements, coal should have a volatile matter in excess of 30 per cent. to be considered as gas coal.

While a great deal of work has been done on the effect of the composition of volatile matter in different grades and ranks of coal, yet it seemed necessary that some form of gasification test is required to integrate the large number of factors that are involved in what appears to be a relatively simple test. Under normal conditions, therefore, we would expect the gas coal to give at least 3000 B.t.u. feet per pound.

The yield of other by-products such as tar and ammonia are of considerably less importance in the consideration of a suitable coal due to the fact that the quantities appear to be dependent to a somewhat greater extent on the particular process of carbonization used than on the original qualities of the coal.

Impurities

The most important impurity, at least from the quality standpoint is the ash. Since the ash all remains in the coke and is, therefore, increased nearly one-half in the coke, the lower the ash naturally the better quality of coke will be produced. Current practice seems to fix the maximum limit at 9 to 10 per cent. ash in the coal.

The effect of the fusion point of the ash has already been discussed. We may point out here that in general the lower the percentage of ash the less the effect of the fusing temperature on the resulting clinker so that unless the coke is to be used for water gas manufacture, the fusion point does not become of great importance.

However, sulfur is one of the important impurities that must be given consideration. Sulfur may exist in the coal in one of a number of forms but when the coal is carbonized, while roughly one-half of the sulfur remains in the coke, the volatile sulfur occurs as hydrogen sulfide and as organic sulfur compounds.

Hydrogen sulfide can be readily removed by existing methods so that the presence of large quantities may result in an increase in the size of the purification plant that will be required and this will be reflected in a slightly increased operating cost. On the other hand, the presence of excessive amounts of fixed or organic sulfur compounds is more serious inasmuch as they are not removed by the ordinary forms of purification used for the removal of hydrogen sulfide and their removal would involve the use of special processes and a considerable increase in purification costs.

While the usual statutory requirements permit 30 gr. of organic sulfur compounds per 100 cu. ft. of gas, modern practice is attempting to remove these compounds as far as possible.

The organic sulfur compounds are composed in a large part of carbon bisulfide with a relatively small quantity (3 to 9 gr. per 100 cu. ft.) of more complicated materials such as carbon oxysulfide, mercaptans and thioethers.

The carbon bisulfide can be removed by treatment with the catalyst at a suitable temperature when it is broken into hydrogen sulfide, but the other organic compounds are much more refractory and usually require special catalysts and relatively high temperatures for their transformation.

It is apparent, therefore, that in many plants, sulfur is a limiting factor in the choice of a coal, yet where there is sufficient price differential to make the process economic, it is proper today to use coals very much higher in sulfur than was formerly considered good practice, but as long as we have the supplies of low-sulfur coals that exist at present, it probably would be a number of years before there was sufficient price differential to result in the use of any large volume of high-sulfur coals.

A. S. T. M. Gas Coal Specification

The general practice of the gas industry in the selection of these coals is well summed up in the specification for gas coal as proposed by the American Society for Testing Materials, which follows:

1. Gas and coking coals must yield both merchantable gas and coke when distilled in a retort or oven by commercial methods. The type of coals may vary within rather wide limits according to the treatment in the retort and the market for the products. These specifications, therefore, merely give the limits within which gas and coking coals will usually fall, and indicate the circumstances under which further restrictive conditions should be imposed.

I. Sampling and Analysis

2. The coal shall be sampled in accordance with the Standard Method of Sampling Coal (Serial Designation: D 21) of the American Society for Testing Materials.

3. Analysis of the coal when required, shall be made in accordance with the Standard Methods of Laboratory Sampling and Analysis of Coal (Serial Designation: D 22) of the American Society for Testing Materials.

II. Chemical and Physical Properties

The carbon ratio, that is, the ratio of fixed carbon to volatile matter, while not entirely reliable, is the best simple index to the behavior of the coal when carbonized The carbon ratio in the gas of gas coals will vary from 1.4 to 2.0, and for coking coals from 1.4 to 5.0. The latter includes a wide range of coals varying from high volatile gas coal to low volatile or "smokless" coal.

4. (*a*) The percentage of moisture in the coal as mined shall be subject to agreement by the purchaser and the seller.

(*b*) In the absence of a definite agreement between the purchaser and the seller, the mine moisture in the coal as mined shall not exceed 4.0 per cent. The moisture shall be determined by the general average composition of coal from the mine in question and an analysis of each shipment shall not be required.

5. The fusion temperature of ash of coal, the coke from which is intended for domestic and industrial use, shall not be below 2200° F. In the case of coke for use in the manufacture of water gas, the fusion temperature of the ash of the coal shall preferably be higher than 2300° F. The fusion temperature of the ash shall be determined in accordance with the method for determination of fusibility of coal ash appearing in the Standard Methods of Laboratory Sampling and Analysis of Coal (Serial Designation: D 22) of the American Society for Testing Materials.

Note.—The fusion point of ash is not usually important for metallurgical work. It is important, however, in the case of coke for domestic and industrial furnace use and for the manufacture of water gas.

Special Requirements for Gas Coals

6. Gas coal shall contain not less than 35.0 per cent. of volatile matter when determined on the moisture and ash-free basis.

Note.—This is equivalent to 30.8 per cent. volatile matter for a coal containing 12.0 per cent. of combined ash and moisture.

7. In the case of gas coals, the ash in the dry coal shall be not over 9 per cent.

8. The composition of gas coal shall be such that the dry coal produced therefrom will contain not over 1.5 per cent. of sulfur and the resultant gas will contain not more than 30 grains of sulfur in the form of compounds other than hydrogen sulfide, per 100 cu. ft. of gas.

9. Gas coal shall be such that the coke produced therefrom will be of sufficient size and strength to be suitable for domestic use or for the manufacture of water gas.

Note.—These physical characteristics of coke are not amenable to simple explicit definition and must necessarily be left to the judgment of experienced operators.

Operating Factors

While these specifications cover what may be called the analytical constants of coal, there are a number of factors that are of some importance from an operating standpoint. The coal should not slag or deteriorate rapidly or be subject to spontaneous combustion on storage as the industry is accustomed to carrying relatively large stores of coal to insure uninterrupted supply of gas.

The effect of the size of coal is still subject to a good deal of discussion in the industry and while considerable experimental work has been done, the question of size is still apparently bound up to some extent with the physical characteristics of particular seams.

In the above discussion we have refrained from mentioning the methods of determining the characteristics but may say that while laboratory methods of examination are of very great service in making preliminary survey, it still appears necessary that the particular coal should be tested on a large scale under commercial conditions before a final answer can be secured as to its suitability.

In the second class of processes for the gasification of coal, *i.e.*, those in which the coal is actually consumed in the presence of air and steam we find that the operating characteristics of the coal in the fuel bed are of primary importance in that since the coal is entirely consumed, all of the heating value must of necessity go into the gas and the quality of the gas is, therefore, rather largely dependable upon the way the coal acts in the fuel bed of the apparatus.

If we restrict this discussion to the use of bituminous coal in water gas generators, we find that the most important condition is the manner in which the coal reacts as it descends toward the heated zone in the fuel bed of the generator. It is necessary that the coal should carbonize readily, should pass through the plastic state quickly and apparently should not swell on heating.

While it is evident that in an intermittent operation, such as the manufacture of water gas, during the blasting periods there will evidently be a considerable loss in heating value of the volatile constituents of the coal. It would be desirable to use a low-volatile coal, yet experience seems to indicate that low-volatile coals tend to go to pieces in the fire and result in clogging of the fuel bed and reduction in the capacity of the apparatus.

Evidently, in operations of this type, the existence of low amounts of ash is of great advantage although the fusing point of the ash does not seem to be as important as where coke is used as a fuel.

Sizing

The sizing of the coal, however, appears to be a very important factor and this is also intimately connected with the hardness of the coal in

that the coal must stand transportation after being sized. Experience appears to indicate that it is absolutely essential to keep all fines and small coal below 2 in., out of the fuel bed and to work with as nearly uniform size lumps as possible.

This requirement has very seriously restricted the choice of coal because many coals will not, owing to their nature, crush to the proper size without a large loss in fines or due to their softness will not stand transportation. It would appear that the industry needs some hardness test for coal that will enable us to apply a quantitative figure to the term "hard."

The question of sulfur in general does not seem to be a very important factor except as it may affect the fusion point and quality of the ash.

In the choice of coals for use in water gas generators, we are therefore restricted to reasonably high-volatile coals that will not decrepitate in the fuel bed and the best results appear to be obtained with the coals that do not swell on heating and which pass quickly through the plastic stage.

Laboratory experiments have not as yet been of very great service and we are again confined to full scale operating tests to determine the true worth of coal for this purpose.

DISCUSSION

J. Keely, Kayford, W. Va.—What is the percentage of sulfur?

W. H. Fulweiler.—I stated that the coal should not give over 30 gr. organic sulfur compounds in the gas.

J. Keely.—How much sulfur is there in it?

W. H. Fulweiler.—The percentage of total sulfur in the coal does not always indicate the quantity of organic sulfur that will be found in the gas.

H. J. Rose, Pittsburgh, Pa.—I think Mr. Keely wants to know the percentage of sulfur in coal which will give, on the average, 30 gr. organic sulfur in the gas.

W. H. Fulweiler.—There is no direct method of determining this. Some of the sulfur remains in the coke and some of it forms hydrogen sulfide. Therefore, simply to state the percentage of total sulfur really does not mean much.

H. J. Rose.—It has been my observation that 1½ per cent. sulfur in the coal will give an organic sulfur content of about 30 gr. per 100 cu. ft. of coal gas. That is only a rough figure and must be substantiated in each particular case. It is generally true that neither gas companies nor blast furnace companies will consider a coal having more than 1½ per cent. sulfur.

W. H. Fulweiler.—Price however is frequently the deciding factor. At one plant we used a coal containing nearly 6 per cent. of sulfur.

Classification from the Standpoint of the By-product Coke Industry

By W. H. Blauvelt,* New York, N. Y.

(New York Meeting, February, 1928)

The only way in which the difficult problems of classification of coal for the manufacture of by-product coke can be solved is to analyze them by the use of scientific data.

It is very easy to adopt classifications for coal for the manufacture of by-product coke along certain lines; we can easily say that for a certain class of metallurgical operation coke made from bituminous coal must not contain more than so much sulfur or more than so much ash and should yield certain returns in the by-products, tar, ammonia, benzol, gas, etc. Those points we are fairly clear on and I think it is hardly worth while to discuss them because I presume that they are familiar to most of us who have any occasion either to make coke or to use it.

But there is a field in the manufacture of coke which we know very little about. For example, we are operating a plant on a certain mixture of coal. Suddenly our blast-furnace man or the user of the coke tells us the coke is not as good as it was before. The coke is analyzed and the coal is analyzed. Everything seems to be about the same as it was—the ash is the same, there is no more sulfur, and so on—but the coke, for the use to which it is put, particularly in the blast furnace, is wrong. What happens? The management decide to try a slightly different coal mixture. They do not necessarily raise the total hydrocarbons, the volatile matter in the mixture, nor change the sulfur or the ash, but they do make a change which improves the value of the coke for metallurgical use. The reason for this difficulty, as I see it, is the "combustibility" of the coke—I hesitate to use the word, but it is the best one I have available.

Combustibility of Coke

We know now that the combustibility of coke is a very vital point in many metallurgical processes and I think one of the most notable instances of the proof of this is the experience of the Carnegie Steel Co. in the production of coke at its Clairton plant. The history of that

* Consulting Engineer.

plant is well known. When the production of coke began there many of the blast-furnace managers of the Carnegie Steel Co. protested. (I am speaking now of a matter of common knowledge rather than making an authoritative statement.) Nevertheless, it appeared that the particular kind of coke produced was on the whole most advantageous to the steel company, and shortly by modifications of the coking process and of the blast-furnace practice remarkable results were obtained. As I analyze the situation, it was due to the fact that although the coke looked almost hopeless to many of the superintendents of the blast furnaces, it did have qualities—we will call them qualities of combustibility for lack of a better term—which made it, after all, an excellent blast-furnace fuel.

We all remember that in the old days, and it is still so in many places in Europe, coke had to be very hard, very dense and strong. Many of those qualities which we used to think were essential are now giving way to a considerable degree at least, to the ability of oxygen to combine rapidly with the carbon of the coke, in the tuyere zone of the blast furnace.

I do not think that this is the time for us to discuss classification of coals for by-product coke oven use on the basis of freight rates or on the basis of sulfur content or ash content, because it seems as if those factors are self-evident, but I do think we have in the qualities of coal mixtures which make certain—shall I call them physical?—characteristics a very important field, a very vital field, perhaps more important and more vital than those which we have been accustomed to regard as routine classification points.

This discussion is very timely because it is related to the discussion at another session of this meeting of the Institute, regarding the specifications for coal for various purposes; that is, what kind of specifications shall the owner of a certain process using coal demand of the producer of that coal? I do not believe that we are ready yet to introduce specifications that will cover the points I have raised and emphasized.

Perhaps in investigations along the lines charted by H. J. Rose, we may find at least part of the answer to the problem of what classification of coal we must ask for from the producer, to give the kind of physical structure in coke demanded for any particular metallurgical use.

R. H. Sweetser has frequently said that every added per cent. of ash costs the pig-iron man 30c. per ton of pig iron. That is true, but when we see the tremendous increase in the capacity of the furnace such as has recently been shown in Germany and in several places in this country, we will find that the physical characteristics, which depend on the scientific classifications, are determined and controlled with equal accuracy and that they are more important than any of the chemical classifications with which we are all familiar.

DISCUSSION

H. J. Rose, Pittsburgh, Pa.—Answering Mr. Blauvelt on the chemical and physical tests for evaluating coke and coal: In my work on the study of coke and coal by ultimate analysis, the ultimate analysis did not tell the whole story in predicting coking value. When we found two coals of different behavior but of apparently similar analysis, we set to work to find out the real differences.

We determined the melting points of the coals, not the melting point of the ash, but the melting point of the coal itself. We also tried to determine the coking point, because we wanted to find out the temperature range over which the coals were plastic. We tried, without much success, to determine just how sticky the coals were, from the time they melted until they coked. Coke formation is a matter of blowing bubbles in a plastic mass. Cell structure and shrinkage depend to a large extent on these characteristics. In many cases we were able to get a good idea as to why two coals of similar chemical analyses showed a different behavior when coked. The results were very encouraging, I think. That sort of work has not been developed far enough to make it generally useful. There are so many variables in coking that I do not know whether we shall ever be able to derive simple rules or formulas for evaluating coking properties, but I can say that the results obtained lead us to believe that we can untangle these many complex variables, one at a time, and find out why coals behave differently. Ten years from now we shall probably know a great deal more about them.

There is one point that is often brought up; that is, why can we not get some simple laboratory test for evaluating the coking properties of coal? Why is it that some test is not of universal application?

Anyone who does very much work on coal carbonization finds that the temperature and conditions under which coal is heated have an enormous effect on the kind of coke produced. I will take an example of that from kitchen technique. Suppose that one puts lumps of bread dough into ovens heated to widely different temperatures. A lump of dough put into an oven that is at just the right temperature for baking bread will become a finished product having the proper cell structure and texture. Dough put into an oven that is not hot enough will keep on rising until it runs over the side of the pan. The same things are true of coal. If coal is heated too slowly, it overflows the container and produces a mass of soft, spongy coke.

On the other hand, a lump of dough put into a red-hot oven would not have time to swell very much before it started to char or carbonize; the "bread" produced would be of an entirely different structure; it would be an entirely different product from that put in an oven of the proper baking temperature, or from bread made in a cool oven. In the same way, depending on the heat in a coke oven, the pressure on the coal, and so on, there will be enormous differences in the structure of cokes produced from the same coal.

That is why we cannot get any one test that will tell how coal will behave under all conditions. We can imitate in a laboratory with some success any particular coking process but we do not have any one test that fits all cases.

W. H. Blauvelt.—Did you ever find it practicable to try a miniature coke oven—for example, an electrically heated chamber somewhat similar in shape to an oven—to see whether you could duplicate the working condition?

H. J. Rose.—We have tried that. We would design something of the kind and then decide that the height or length should be greater, and before we got through we would decide that we wanted a full-size coke oven.

We do successfully use a laboratory test on quantities of coal as small as 2 oz. when we do not have a large quantity of coal available. In this test we produce a

coke that is fairly equivalent in cell structure to the coke that would be produced in the by-product oven. From experience we have found that by imitating the progressive heating in a coke oven, at the proper temperature, we get a coke from which we can deduce the coking qualities of coal. However, it is very difficult to produce coke on a small scale which is closely equivalent to coke made in regular ovens.

One can sit down with paper and pencil and figure out an ideal laboratory apparatus, but it would be too complicated. The flow of heat in the coke oven requires from 12 to 18 hr. To imitate that, it is necessary to have two or more shifts of chemists regulating heating controls. These tests can be made according to elaborate plans, but when they are done one wishes that he had results from a regular coke oven.

There is no theoretical reason why oven conditions cannot be imitated, but since we have coke ovens available, we have always preferred to make full-scale tests. They satisfy everybody, both the chemist and the practical man, and do not cost much, where the ovens are available.

Classification of Coal from the Standpoint of the Steam Power Consumer

By S. B. Flagg,* New York, N. Y.

(New York Meeting, February, 1928)

Advancement in the art of burning fuels for steam generation has been so marked and so rapid in the last 10 or 15 years that one may well hesitate to classify as unsuitable for stationary steam boiler firing any material having a fuel value. This statement should not be interpreted to mean that any existing boiler plant can successfully utilize any fuel with which it might be supplied. It does mean, however, that almost without exception it is possible to design and construct stationary boiler plants so that they can be operated with whatever coal is the economic supply therefor.

In considering the subject of the use classification of coals, it must be recognized, first, that the suitability or unsuitability of a coal for certain uses or conditions may be largely a matter of opinion and, second, that as advancement in the art of burning fuels continues, fuels that may have been considered unsuitable may later be regarded as entirely suitable. This is well illustrated in the creation of a market for slack bituminous coal and for the finer sizes of anthracite that resulted from the development of mechanical stokers adapted thereto. Coke breeze is another fuel of which large tonnages were formerly wasted, that has now been found suitable for use in properly designed furnaces. Again, results that would have been thought quite impossible a generation ago are being obtained in steam boiler plants today with lignites of high moisture content and with coals of excessive ash content.

It must, however, be admitted that oftentimes an existing boiler plant can be successfully operated with only a limited number of the coals that are available as possible supplies. We may briefly consider some of the more important factors that may affect the suitability of coals for such a plant. Insufficient or barely sufficient draft to meet peak steam demands is often a limiting factor, and the permissible frequency of fire-cleaning periods is another. In some cases, the action of the molten coal ash on refractory furnace linings prohibits or makes inadvisable the use of certain coals. Again, clinker conditions in the fuel bed or slag conditions on the boiler-heating surface may be much worse with some coals than with

* Electric Bond & Share Co.

others. Labor conditions in the plant or in the community must sometimes be given considerable weight in determining what is the proper coal to select. Obviously, the furnace design, as well as the type, size and condition of the fuel-burning equipment, will be factors, and they may be the chief ones affecting the suitability of a coal. Bearing in mind that any number of these factors may enter into the problem of fuel selection for a specific plant, we may outline in a general way for different types of fuel-burning equipment commonly used in steam power generation what should be the characteristics of coals that may be called suitable therefor.

Hand-fired Furnaces

For hand-fired boilers, any coal from an anthracite to a lignite, or even peat, must be tentatively classified as suitable, but differing furnace designs would be provided for the different classes of coals. There must, of course, be the proper interrelation between draft, size of coal and steam demand. In view of the labor of hand-cleaning of fires, a low ash content coal is preferable but under certain conditions an ash content of even 25 to 30 per cent. may be permissible. The higher the ash-fusing temperature, the less will be the clinkering troubles. Ordinarily, serious clinker difficulties should not be encountered if the ash-fusing temperature is above 2200° F., but increasing degree of trouble may be expected as temperatures fall below this figure. However, coals with ash-fusing temperatures as low as 1900 °F., are being used and may be the proper selections under some conditions. In communities where smokeless or comparatively smokeless operation is a necessity, high-volatile bituminous coals will not be suitable unless the furnace has been designed for such coals and the firing is well done.

Stoker-fired Furnaces

Traveling grate mechanical stokers are successfully burning a wide range of coals, and for the firing of boilers so equipped, most of the coals from lignites to anthracites may be considered suitable, with the exception of low-volatile semibituminous coals. However, furnace designs adapted to the particular type of fuel to be burned must be provided, if satisfactory results are to be obtained. With this type of stoker, coals of high ash content—even up to 30 per cent.—may often be utilized without serious operating difficulty; in fact, trouble is more likely to occur with coals of very low ash content as the refuse therefrom affords inadequate protection for the grate surface and the links or keys usually overheat. The success of this type of stoker in burning coals of low ash-fusing temperature has been conspicuous, so that practically no limitation need be imposed in this respect.

The underfeed stoker-equipped boiler plant may utilize semianthracites, semibituminous or bituminous coals, or lignites if the furnace has been properly designed for the selected type of coal, although this type of stoker was originally designed for and is best adapted to caking coals of moderate or low ash content. Installations are in operation, however, where coals of relatively high ash content with ash-fusing temperatures somewhat below 2000 °F., are being successfully burned. Naturally coals with less fusible ash are less likely to give trouble than the type just referred to. Size of coal affects suitability somewhat, especially with coals of the semibituminous type. Too high a percentage of finely disintegrated coal limits the boiler output and also affects efficiency.

Powdered Coal

Before much experience had been gained in the burning of coals in powdered form in boiler furnaces, it was a common opinion that the adoption of this method of combustion in steam generation would broaden the list of available and suitable coals in much the same way that the mechanical stoker had changed the situation as compared with hand-fired conditions. Such has not proved to be the case, however, and especially when the economic aspects of coal selection are considered, it may in many cases prove advisable to use only the better grades of the available coals. For instance, a lower grade coal that may figure out cheaper on the basis of delivered B.t.u. per cent. may actually cost as much or more per million B.t.u. when the cost of preparation is taken into account. Although it is a fact that almost every type of coal from anthracite to lignite is being burned in powdered form in boiler furnaces, the designs of these furnaces must be worked out more thoroughly and exactly if successful operation is to be assured, than in the case of stoker-fired furnaces. The reason for this may not be apparent at first but will be more readily perceived when one remembers how easily and instantaneously the rate of coal feed to a furnace may be increased or its temperature raised. Changes in powdered fuel furnaces take place much more quickly than in stoker-fired or hand-fired furnaces and the designer must take full account of this fact. The use of water-cooled surfaces for the sides or bottom of powdered coal furnaces has shown a marked increase, as only thereby can the probability of operating troubles from molten ash be avoided. Even with such water-cooling, some engineers have preferred to arrange for melting and drawing off periodically in molten form the ash that accumulates in the furnace bottom, than to endeavor to prevent the fusing.

Summary

Summarizing the matter for pulverized fuel furnaces, it may be said that although practically every type of coal can be burned, the difficulties

of getting satisfactory results generally increase with the increase of fixed carbon content and with the decrease of volatile matter content. If the furnace is to be operated at a high rate of heat evolution, it will usually be advisable to have a coal with ash-fusing temperature of not less than 2300° F. Furnaces without water-cooling surfaces may even require coals of that type or with an even more infusible ash to operate at moderate rates of combustion unless high percentages of excess air are used. It should be noted, however, that the type of burner used and the method of admitting air for combustion, as well as the temperature of this air, to a large extent affect the suitability of certain fuels.

From what has been stated in the foregoing paragraphs, it is evident that no hard and fast rules can be formulated for a use classification of coals for steam generation. If the limitations of an existing boiler plant are sufficiently well known, however, and the principal characteristics of the available coals can be accurately ascertained, one can with a fair degree of certainty classify these coals as to their suitability for use in that plant. In the case of a new boiler plant, the problem is one of studying the characteristics of the various fuels available and deciding in view of their comparative delivered costs which will probably be the economic supply; then the station design must be worked out so as to give with this fuel not only efficient operation but a minimum of operating troubles.

Classification of Coal from the Standpoint of the Coal Statistician

By F. G. Tryon,* Washington, D. C.

(New York Meeting, February, 1928)

This paper treats only of the practicability of introducing a standard classification into the records of production and distribution of coal which we try to keep in the Bureau of Mines.

From the point of view of gathering and analyzing statistics, the requirements of a satisfactory classification are that it shall be so definite as to eliminate the factor of arbitrary judgment, that it shall be reasonably simple and, finally, and most important, that it shall be understandable to the trade. In other words, to be woven in the statistical record it must be adapted to the raw material which flows in from the 7000-odd mines that produce the coal. The success of any scheme of classification, from the point of view of use, will depend largely on the degree to which it utilizes the accumulated experience of the trade and employs as far as may be possible the accepted trade designations. The experience of the Tidewater Coal Exchange during the war would be an excellent point of departure for a use classification of the coals of the Eastern Appalachians. Although the exchange long since passed out of existence, coal is still quoted and sold by the old pool numbers, and they have shown rather extraordinary vitality.

Some years ago, in the Geological Survey, the experiment was tried of getting operators to classify their own coal by offering them a list of standard designations on the annual statistical questionnaire. The results were more amusing than informative. What with the confusion in terminology and the natural temptation to put one's best foot foremost, there was a general tendency to grade up the quality. Floods of semi-bituminous coal flowed in from many sources, millions of tons from districts where not a pound of that quality has ever been produced. The producers of semianthracite in Arkansas almost without exception would report their product as anthracite and their neighbors in the little Paris field, producing a fine blocky house coal averaging about 17 per cent. volatile, would say, "Our coal isn't anthracite; it's a semianthracite." Finally we discontinued the publication of those data as being more misleading than helpful.

* Head, Statistics Section, Coal Division, Bureau of Mines.

The alternative was to try to impose a classification from above, and the lack of agreement even among the scientists made this inadvisable. Our present attitude is one of watchful waiting and expectancy. If and when a classification can be devised which will be generally accepted (we shall have to make that a condition) we will introduce it in the statistical record with great joy.

In the meantime, we can carry along certain specific tasks which point in that direction. One of them is to make more precise the basis of geographical classification which we now employ.

Classification by Counties

At present we limit ourselves largely to giving data by counties. A county can, of course, be exactly defined. The defect of the county classification is that it frequently ignores geological conditions. In Fayette County, West Virginia, to take a conspicuous example, the coal is partly high volatile and partly low volatile and the statistics for the county are a composite of sharply differing coals. To get around that difficulty we have begun to regroup the figures by districts, showing for example the New River and Kanawha fields, but even this is unsatisfactory because the district boundaries are often drawn quite as much with reference to labor conditions or freight rates as with reference to the quality of the coal.

In the same way it would be possible to group the statistical records according to some standard map, such as the isovol map of Pennsylvania prepared by Sisler,[1] which classifies coals by volatile matter. With the map as a base we could show the production of mines lying between specified isovols.

More important in building for the future is the card record which we keep of the production of the individual mines. We have a card for every mine in the country giving data which run back about 20 years, and it is possible to throw those cards into a new grouping; so when the classification arrives at the point where it can be applied, it will be feasible to allocate these tonnages by classes and show not only the future but enough of the past production to indicate the rate of growth and the trend.

Statistics on Sizes

Finally, we can do something with the related question of sizes. Our friends in the anthracite region find that of all the material published by the Bureau perhaps the most useful is the relative percentage of each size shipped and the average sales realization on each. So far nothing has

[1] J. D. Sisler: Volatile Matter in Pennsylvania Coals. Penn. Topographic and Geol. Surv. *Bull.* 81.

TABLE 1.—*Percentages of Prepared Sizes, Run-of-mine and Slack or Screenings Produced by Bituminous Coal Mines in the Months of August–December*, 1917

[From an unpublished study prepared by David L. Wing of the Federal Trade Commission for the use of the Engineers Committee of the U. S. Fuel Administration. The production of the mines reporting was 191,181,627 tons or 82 per cent. of the total production for the country during the same period.]

U. S. Fuel Administration Price District	Prepared Sizes, Per Cent.	Run-of-mine, Per Cent.	Slack or Screenings, Per Cent.
Arkansas:			
Anthracite	51	4	45
No. 1	21	65	14
No. 2	80	4	16
State Total	30	49	21
Colorado:			
Anthracite	58		42
Domestic	53	26	21
Lignite	43	37	20
Trinidad	20	48	32
State Total	40	36	24
Georgia, State	28	1	71
Illinois:			
No. 1	67	19	14
No. 2	61	16	23
No. 3	55	27	18
No. 4	42	41	17
No. 5	38	54	8
No. 6	47	34	19
State Total	50	31	19
Indiana:			
Bituminous	43	36	21
Block	83	1	16
State Total	44	35	21
Iowa:			
Albia	63	14	23
Appanoose	95	2	3
Des Moines	62	14	24
Northern	86	5	9
State Total	70	11	19
Kansas:			
Leavenworth	77	4	19
Osage	73	15	12
Pittsburgh	52	19	29
State Total	54	18	28
Kentucky:			
Blue Gem	36	47	17
Eastern Kentucky	37	46	17
Elk Horn Thick Vein	19	77	4
Jellico	35	43	22
Southeastern	24	68	8
West Kentucky	36	47	17
State Total	32	54	14
Maryland:			
Cumberland-Piedmont	2	98	
Georges Creek	12	88	
Upper Potomac		100	
State Total	4	96	
Michigan:			
State of Michigan	37	40	23
Missouri:			
Bevier	68	11	21
Lexington	88	10	2
Linn-Putnam County	19	75	6
Novinger	72	3	25
Pittsburgh	61	12	27
State Total	69	15	16
Montana:			
Bituminous	50	37	13
Lignite	83	6	11
State Total	50	37	13
New Mexico:			
Carthage	11	89	
Cerrillos	51	8	41
Gallup	71	9	20
Raton	45	15	40
State Total	48	16	36

TABLE 1.—(*Continued*)

U. S. Fuel Administration Price District	Prepared Sizes, Per Cent.	Run-of-mine, Per Cent.	Slack or Screenings, Per Cent.
North Dakota:			
State of N. Dakota	59	33	8
Ohio:			
No. 1	38	48	14
No. 2	25	62	13
No. 3	30	61	9
Pomeroy	52	27	21
State Total	37	49	14
Oklahoma:			
No. 1	17	76	7
No. 2	58	26	16
State Total	24	68	8
Pennsylvania:			
Central	4	91	5
Southwestern Pennsylvania	31	56	13
State Total	20	71	9
Tennessee:			
Blue Gem	33	50	17
Eastern Tennessee	22	68	10
Jellico	38	43	19
West Tennessee	16	63	21
State Total	20	64	16
Texas:			
Bituminous	98		2
Lignite	31	64	5
State Total	52	44	4
Utah:			
State of Utah	39	27	34
Virginia:			
Anthracite	57	29	14
Black Mountain	44	34	22
Pocahontas	37	29	34
Southwestern	10	75	15
Upper Clinch Valley	20	67	13
State Total	12	73	15
Washington:			
No. 2	84	11	5
No. 3	32	68	
Roslyn	6	84	10
State Total	44	49	7
West Virginia:			
Coal and Coke	21	69	10
Fairmont	23	64	13
Gauley		100	
Kanawha	23	63	14
Kenova	45	30	25
Logan	25	58	17
New River	5	88	7
Panhandle	44	37	19
Pocahontas	6	82	12
Pomeroy	32	63	5
Putnam	76	1	23
Thacker	26	57	17
Upper Potomac	3	91	6
No. 10	19	65	16
Tug River	12	76	12
State Total	15	73	12
Wyoming:			
No. 1	55	35	10
No. 2	11	83	6
No. 3	40	51	9
State Total	30	62	8
Total United States	32	54	14

been done on the relative sizes of bituminous coal and still less on the values they bring. Beginning with 1927, we expect to get figures showing the total run-of-mine, prepared sizes, and screenings shipped from each district, and later hope to link with these the values obtained so as to show the average price on each size. These figures should be of interest to the trade. A statement (Table 1) giving for each field in the country the relative proportions of the different sizes produced during a certain period of the war was worked up for the Engineers Committee of the U. S. Fuel Administration in connection with the fixing of maximum prices. The result for the United States showed 54 per cent. run-of-mine, 32 per cent. prepared sizes, and 14 per cent. slack or screenings.

In closing, let me say that we are most anxious to receive suggestions from the industry as to the possible grouping of the statistical data. For example, if the coke producers and coal associations can agree on a list of mines producing coal suitable for by-product coke, we shall be happy to translate the list into a statistical statement.

Closer Cooperation between Scientists and Practical Men

DISCUSSION

(New York Meeting, February, 1928)

W. H. Blauvelt, New York, N. Y.—One thought has been running through my mind during the whole of this meeting and that is that the scientific and the practical men must recognize very clearly their interdependence. The value of scientific classification of coal is going to be of the very greatest importance to the users, the practical men. Is not the scientific investigation and the great amount of work that is being done in a measure at least lost to the world?

Therefore the scientific investigations and the practical needs of the users should be brought as close together, and the work should be done as closely together, as possible; that is, the scientific man should know what the practical man needs, and the practical man should not feel that the scientific man is merely a theorist.

Scientific investigation is going to have a great effect and be of great value to the practical man who is trying to solve his problems. In the subject which I discussed,[1] for instance, we must look to the scientific man to tell us why coal cokes in one way and not in another and why certain coals will do certain things and others, which appear on the surface to be generally similar, do not act in the same way.

Need of Physical Data on Coal

H. J. Rose, Pittsburgh, Pa.—There is a subject which is a sort of hobby of mine; that is, the need for what might be called critical tables of constants for coal.

The chemist has big volumes giving constants, such as the density, solubility, specific heat, thermal conductivity and all other physical and chemical data that have been determined for chemical compounds. Data are also available on many common materials, for example, the density, hardness, modulus of rupture or maximum crushing strength of swamp white oak as compared with some other wood. In fact, physical data for practically every type of wood commercially produced in the world can be found. In the same way the metallurgical man has handbooks giving the properties of alloys, etc., but we have very little tabulated information about coal.

Our principal reference works are the volumes published by the U. S. Bureau of Mines on the proximate and ultimate analysis and calorific values of coals of the United States. That is a pioneer work of enormous value, but we have reached the point where we need more diversified information. Mr. Fulweiler raised the question of the relative strength of the different coals and cokes. Dr. Campbell and Dr. Ashley wanted to have some information on the resistance of coal to air slacking. It was brought out in the discussion that we want to know something about the coking properties of coal. There are any number of physical tests that we need, such as the thermal conductivity of certain coals, the coefficient of expansion, or the coefficient of friction; or other constants. A smattering of such information is scattered through the literature; for instance, a research worker in Wales will report a coefficient of expansion on Welsh coals and one in South Africa will report something else for one or two kinds of African coals, but it is difficult to find the information that is needed on our own coals. Would it not be worth while to consider collecting reliable critical

[1] See page 473.

data in some convenient form? The information would have to be carefully selected. We could not include everything we have because it would be a hodgepodge.

The Bureau of Mines in its early work published a couple of papers on the volatile matter of coal, which are a mine of information. Perhaps we could take fresh samples of the same coals and make every chemical and physical test that we know, and publish the combined information (with the methods used) as an example of what can be done, and as an inducement to others to make similar determinations on coals in which they are interested.

E. McAuliffe, Omaha, Neb.—As a so-called practical man I want to express my satisfaction in the scientific papers read and discussed this morning. I have been charged with being overcritical of the coal industry; as saying that we are definitely lacking in technique and that we are not sympathetic with the scientific work that has been extended to us from time to time. In the matter of mine explosives, without question, the research work conducted under the direction of the U. S. Bureau of Mines, has saved thousands of lives in American mines.

We cannot, of course, get away from the dollar and cent side. If we were more sympathetic with the work of the scientists we undoubtedly would earn our dollars more easily and perhaps earn more of them. I hope that I will not fail to express my appreciation of the work that has been done by the scientific men when opportunity offers. Our industry is distinctly behind other industries that are quite as difficult of management and control. The steel industry has carried on much research work, so has the oil industry and many of the manufacturing industries. They have all gone far beyond any point we have reached, and yet we have some tremendously elusive questions yet to be answered.

Volume of Fuel Research

A. C. Fieldner, Pittsburgh, Pa.—It is true, as Mr. Rose points out, that fundamental research work on coal is conducted on far too small a scale in this country. During the last two years, I prepared the chapters on "Coal, Coke and Gaseous Fuels" for the Annual Survey of American Chemistry, published by the National Research Council. In preparing this chapter, I had occasion to make an estimate of the relative amount of fuel research conducted in America, Great Britain, Germany and France. In order to obtain some idea of its amount, I took as a criterion the number of papers from each country containing experimental data on the subject of coal (including lignite and peat) and its products—gas, coke and by-products—given in the Fuel Section of *Chemical Abstracts* for the calendar year 1926. The results of this study were as follows:

Country	Total Number of Papers for 1926	Total Population of Country, Millions*	Number of Papers per Million Population
United States	104	117	0.88
Germany	103	62	1.66
Great Britain	85	36	2.36
France	31	39	0.80

* 1926, according to *World Almanac* for 1927.

If we accept this criterion on a per capita basis the United Kingdom leads in coal and gas research, with Germany next and America and France considerably in the rear.

A further comparison with regard to the nature of the papers shows that the European countries are in the lead of the United States in fundamental research on

the constitution of coal and in the processing of coal, such as low-temperature carbonization, liquefaction of coal and the synthesis of chemical products such as methanol, ammonia and hydrocarbons.

Fundamental research on coal has been limited practically to the laboratories of the U. S. Bureau of Mines, the U. S. Geological Survey, the Canadian Department of Mines, the University of Illinois, and a few other isolated workers in university and industrial laboratories. I must admit frankly that the amount of fundamental research on coal conducted by the Bureau of Mines has been steadily decreasing since the Bureau first entered this field. This is due to the fact that the amount of money available for this work has remained the same ever since its inception. With the constantly decreasing purchasing power of the dollar and with the increasing salaries required for research work, the amount of work now done is about half that carried on 15 years ago. Also, the Bureau's fuel-testing appropriation was made for the analysis and testing of coals belonging to and for the use of the United States Government, consequently the major part of it is being absorbed by this constantly growing routine work for the Government itself.

Early Work of U. S. Bureau of Mines

In the early days of the Bureau, the amount was large enough so that a considerable proportion could be used in a broad interpretation of the act. Dr. Holmes, the first director, considered it essential to have first of all a knowledge of the constitution and properties of the various coals of the United States, in order that they might be utilized most efficiently. The constitution of coal was attacked by three different methods and the organization formed to conduct these researches was headed by a triumvirate consisting of Dr. J. C. W. Fraser, who undertook the study of the constitution of coal by methods of resolution by solvents, Dr. H. C. Porter, who attacked the same problem through a study of the volatile matter in coal and the products of destructive distillation, and Dr. Reinhardt Thiessen, paleobotanist, who studied the origin of coal and its constitution with the use of the microscope and other means.

Dr. Thiessen is the only surviving member in the Bureau of this group. His work has been of outstanding value and he is a recognized world leader in his field. I believe that if the other two investigators had been able to continue their particular lines of investigation with adequate support within the Bureau, that we would now be occupying the leading position in research on coal constitution which we must grant to Dr. Wheeler and his organization.

Dr. Thiessen has carried on his work in the face of considerable disappointments. It is only recently that industry has begun to recognize the value of his research. Practically all of his work was done without any assistance. During the last few years, "fellows" have occasionally done their graduate work under Dr. Thiessen's direction. These fellowships have materially increased his output. However, he has ideas enough to keep half a dozen men going. I wish that some means were available to place more research assistants at his disposal.

H. J. Rose.—I believe every coal technologist will agree that those early papers put out by the U. S. Bureau of Mines on such matters as the volatile matter of coal have been of great and permanent value. Only yesterday a technical man who had a serious problem in coal carbonization asked me some pointed questions. I said, "If my memory is correct, you will find in *Technical Paper* No. 1, the exact information you want regarding the composition of the volatile matter coming from coal at various temperatures." He was prepared to spend thousands of dollars for that information and it was all there for him. Time and time again we have gone back to those papers and have found information that was useful. It is a matter of great regret to me that the Bureau of Mines has been unable to keep up the production of that high-grade basic work on the properties of coal.

Natural Groups of Coal and Allied Fuels

By Marius R. Campbell,* Washington, D. C.

(New York Meeting, February, 1930)

Coal is the geological product of entombed vegetal tissues. This view of its origin led Stopes and Wheeler to define it as "mummified plants." They evidently intended this term to be used in a broad way and to mean the preservation of the organic material, as such, regardless of the means employed to accomplish the purpose. The term "coal" should be regarded, therefore, as broadly generic, and applicable to all accumulations of vegetal tissues as soon as they are covered and protected either by water or by earthy material. In this sense peat should be regarded as the first stage of coal formation and the term coal should include the resultant products of transformation in all other stages of the process until they have lost all resemblance to the original matter from which they were derived.

If this idea of the development of coal is accepted and it is admitted, as the geologists claim, that the forces which produced this transformation of the coal have been active since vegetation began to grow upon the land, and are still in operation, it is obvious that somewhere in the world there are all gradations of this material from peat just forming in the swamps of today to graphitic anthracite which presumably marks the most advanced stage that can be recognized with certainty as belonging to the series. Some of these coals, particularly those occurring in the upper part of the series, are fairly well known because they are the ones that are most widely distributed and have been mined and utilized most extensively. On the other hand, the coals in the lower part of the series are but little known because they are not so abundant and, owing to their relative inefficiency, have been only slightly exploited and utilized, especially in regions where the better coals are abundant. The result of these conditions is that much research work has been done on coals of the higher ranks and a great body of facts regarding their physical and chemical properties and the uses to which they are best adapted has been accumulated, but little information is available regarding the coals of the lower ranks.

Many able scientists, particularly chemists, have attempted to bring some order of the chaos of coals of various compositions and uses by

* Published by permission of the Director of the U. S. Geological Survey.

dividing them into groups, but so far none of these propositions has been entirely successful. The question naturally arises as to why they have not succeeded, and work of the present committee on the classification of coal would be much facilitated if the reasons for past failures were clearly understood. As the writer views the situation, the failure of the previous propositions for the general classification of coal may be listed under five heads, as follows:

1. Failure to recognize that rock metamorphism is the cause of most of the characteristic differences in coal and, therefore, that this should be the prime factor in its classification.
2. Lack of understanding of the physical and chemical properties of the low-rank coal, and lack of differentiation of the properties that presumably will always affect their transportation and use from those which have merely a transient value.
3. Limitation of a scheme of classification to a part only of the general series.
4. Basing a classification on a scheme so complex that it can be understood only by an expert chemist or mathematician.
5. Indifference of the producers of coal, the dealers in coal, and the general public to the need of a standard classification.

Materials to Be Classified

We should profit by the mistakes of the past and realize that it is hopeless to undertake to classify things until we know what those things really are. We must not only study the materials themselves, but we should study the relation of one kind of material to another to see if there are not some natural resemblances and differences which will serve as a basis for the establishment of groups and also for the arrangement of the groups into a logical sequence that agrees with the progressive transformation of the material. In doing this we must disregard, to a certain extent, present usage, because usage is a variable factor, changing from day to day and year to year; but we must look for inherent properties which, so far as we can determine, will always affect the value of the coal, no matter how it may be used.

We must remember that any division we attempt to make into classes or groups is artificial. Nature made no such divisions, hence we shall constantly encounter difficulties in separating one coal from another, but even though divisions are arbitrarily made and may be subject to severe criticism, it is better to make the attempt than to sit idly by and say that because a classification is difficult to make and defend it should not be undertaken. Any reasonable one is better than nothing, provided it is simple and can be understood by those who use it, or are affected by it.

It is the writer's belief that we need to recognize divisions having three orders of magnitude, as follows: (1) Major divisions called groups; (2) minor divisions of the groups into classes: (3) a subdivision of classes into types. Such a threefold division enables one to show not only broad resemblances and differences but also those of lesser magnitude which pertain only to subordinate classes or types.[1] This gives a classification which is much more useful than one based entirely on divisions of a single order of magnitude.

Division into Groups and Classes

In accord with the idea expressed above, the writer proposes to divide all coals into four great groups; some have long been recognized but others are new, or are old groups or classes rearranged and given new names. In this paper the treatment of all groups and classes will be in the order of their development from lower to higher ranks, beginning with peat and ending with the most highly metamorphosed forms that have been recognized. The four great groups mentioned are: (1) browncoal group, (2) hydrobituminous group, (3) bituminous group, and (4) anthracite group.

Browncoal Group

Under this head the writer would include all of the earlier stages of coal formation, beginning with peat and ending with the most highly developed form of lignite with which we are familiar. All of these forms are characterized by three features which link them together into one great group. These features are: (*a*) extremely high percentage of moisture, ranging from about 90 per cent. in peat down to about 27 per cent. in high-rank lignite; (*b*) generally brown color; and (*c*) their lack of the physical properties of what may be regarded as a true coal; hence they represent the incipient stage of coal formation. The term "browncoal" is selected as the name of the group because this color is more or less characteristic of the elements composing it, and also because this term carries no implication as to the chemical or physical properties of these elements. In the United States only two classes of this kind of coal have been recognized—peat and lignite—but it is maintained by many that the German *braunkohle* (browncoal) represents a phase intermediate in character between them. If this is true it should be recognized in any general scheme of classification. On this assumption the browncoal

[1] The term "type" has been adopted by the Committee on Coal Classification to designate certain coals of which the distinguishing characteristics are due not to metamorphism but to some peculiarity in the vegetal matter from which the coal was derived.

group should be divided into three classes as follows: (*A*) peat; (*B*) German browncoal; and (*C*) lignite.

Class A—Peat

As the process of coal formation begins when vegetation falls and is covered by the acidulated water of a swamp, the first class of the browncoal group is peat, which is unconsolidated vegetal tissues containing from 80 to 90 per cent. of water. When peat is dried and macerated for use as a fuel it can with difficulty be distinguished from woody lignite.

Class B—German Browncoal

As the writer is not familiar with this material he will not attempt to describe it or to assign it a definite place in the orderly metamorphism of vegetal tissues into coal. It may or may not deserve to be considered as a distinctive class in this group.

Class C—Lignite

The writer does not believe that the term "lignite" is an appropriate one to apply to this class, because the term means simply "woody," and the coals of this class are no more woody than many others, and those that are included in it are not prevailingly woody. He is using the term because of its common application to coals of this class and because he does not know of a better term to supplant it.

Lignite is simply a peat that has been compressed, mainly by the weight of superincumbent earthy material, which has been deposited on the peat swamp; consequently it differs little from the material from which it was derived. It is generally brown in color; in places it consists entirely of logs of wood and in others of a fat waxy mass which has much the appearance of dark clay. The former is the woody lignite type and the latter is the incipient cannel type. The brown color holds until the moisture content is reduced to about 27 per cent. and beyond this point there is a gradual transition into the glossy black coal of the next higher group. Lignite slacks when exposed to the weather and will readily oxidize and take fire spontaneously.

Lignite occurs in abundance in North Dakota, South Dakota and eastern Montana, the beds in places being as much as 35 ft. thick. It is present in untold amounts on the Gulf Coastal Plain in Texas, Arkansas, Mississippi and probably in Alabama. It is mined commercially in the Dakotas, Montana, Colorado and Texas. It also occurs sporadically in other parts of the country, but in limited areas which at present have no commercial value.

The chemical composition of lignite of this country is shown by the analyses in Table 1, which represent it in the as-received condition. The table also shows the moisture B, and heating value B recalculated to the standard condition of 8 per cent. of ash. These analyses show that there is a close relation between the heating values and the moisture content of the lignites, the only exceptions being Wootters and Hoyt of Texas and Ione of California. These are either incipient cannels or are intermediate in composition between the woody lignites and the cannels. Because of the greater heating values of these high-volatile coals and also because of the probability that they will be devoted to special uses in the future, it is desirable that they should be recognized as a distinct type of the lignite class.

TABLE 1.—*Analyses of Lignite Coals*

A, As-received Basis; B, 8 per cent. Ash Basis

	Moisture		Volatile Matter	Fixed Carbon	Ash	B.t.u.	
	A	B	A	A	A	A	B
Hebron, N. D.	27.8	28.6	31.4	30.2	10.6	7150	7360
Beulah, N. D.	34.3	33.3	26.9	33.5	5.3	7460	7250
Wootters, Tex.	33.5	34.5	39.5	16.3	10.7	7140	7350
Glendive, Mont.	34.9	34.9	43.5	13.5	8.1	6980	6990
Buick, Colo.	35.0	36.0	26.3	28.2	10.5	6650	6840
Hoyt, Tex.	36.8	36.1	28.9	28.1	6.2	7100	6960
Lester, Ark.	39.5	41.6	25.3	22.6	12.6	5880	6190
Williston, N. D.	44.6	43.0	24.3	26.5	4.6	6120	5900
Opheim, Mont.	45.8	45.0	22.5	24.6	7.1	5470	5420
Ione, Calif.	45.8	45.6	30.8	15.8	7.6	6060	6030
Reform, Miss.	48.8	47.7	24.2	21.2	5.8	5570	5410

HYDROBITUMINOUS GROUP

This term, if one may use such a hybrid, is proposed for the great group of coals next in rank in above the browncoal group. If one studies carefully the analyses of the low-rank coals he will see that a great moisture content is their distinguishing feature and that this element decreases in amount gradually from peat at nearly 90 per cent. until it reaches about 5 per cent. near the base of the so-called bituminous coals. Above this point, moisture, although always present, is in such a small amount as to be negligible in the classification of coal into groups and classes. The coals in the lowest group characterized by high moisture are generally known as lignites or browncoals because of their lack of metamorphism, consequently in the naming of the group the term browncoal was retained as being best known and thoroughly distinctive. The coals lying next higher in the series resemble lignite in their high moisture

content, but in their physical properties are closely related to the coals usually designated as bituminous. Therefore it seemed to the writer advisable to select some new term to express the fact that these coals are closely related to, if not an integral part of, the bituminous coals, and at the same time to imply that they contain an unusually high percentage of moisture. The term "hydrobituminous" seemed to satisfy both of these conditions, consequently is proposed for them. As thus defined, the hydrobituminous group contains two clearly differentiated classes; a lower class characterized by a high moisture content and rapid slacking of the coal when exposed to the weather, and an upper class which is only slightly affected by the weather and is closely allied to the coals of the bituminous group above.

Class A—High-moisture Coal

This class includes the coals next higher in rank than lignite, or Class C of the browncoal group, consequently its base must be the same as the upper limit of the lignite class which, as given on a previous page, has a moisture content of 27 per cent. The upper limit is not so easily determined, as the slacking property dies out gradually upward as the moisture content diminishes. A careful comparison of many analyses, aided by a personal knowledge of these coals in the field, has led the writer to believe that slacking ceases to be an objectionable feature at the point where the coals have a moisture content of 13.5 per cent. He realizes that more data coupled with a personal examination of the coals now being mined in the debatable districts might lead to a change of a few per cent., but does not believe that the change would seriously affect his general conclusions.

The coals of Class C of the browncoal group pass by insensible gradations from brown to black into Class A of the hydrobituminous group. The line already described as separating them is an arbitrary one but, so far as the writer is aware, does not do injustice to any coal. The upper limit of Class A of the hydrobituminous group is not so easily settled, as more of the coals near the proposed dividing line are on the market and any change in their classification might affect them seriously. Because of that fact the upper limit of Class A should be carefully considered. Analyses of some of the better known coals of Class A are given in Table 2. In this table the moisture is given in the as-received form and also as recalculated on the basis of all coals having an ash content of 5 per cent. In the latter form the moisture content ranges from 10.5 to 30.8 per cent. All of these coals have been classed by the U. S. Geological Survey as subbituminous, except the coal from Superior, Wyo., which is undoubtedly a bituminous coal—as this term has been applied in the past. In other words, all of the coals listed, except that from Superior, will slack on

exposure, whereas the Superior coal does not slack under any ordinary condition. If the upper limit of Class A were drawn at a moisture content of 10 per cent., the Superior coal would be assigned to the lower class whereas it is universally regarded as superior to the other coals listed.

TABLE 2.—*Analyses of Coals In or Near Class A, Hydrobituminous Group*
(A, As-received Basis; B, Ash Content of 5 Per Cent.)

	Moisture		Volatile Matter A	Fixed Carbon A	Ash A	B.t.u.	
	A	B				A	B
Hanna, Wyo	10.5	10.5	40.3	44.1	5.1	11,210	11,220
Gallup, N. M	10.5	10.5	37.6	47.0	4.9	11,930	11,920
Mt. Harris, Colo	11.0	11.1	37.7	45.7	5.6	11,455	11,570
Superior, Wyo	13.1	12.9	35.8	47.7	3.4	11,619	11,430
Coalville, Utah	13.7	13.6	38.3	43.6	4.4	10,870	10,800
Gebo, Wyo	13.8	13.9	35.5	45.5	5.2	11,210	11,240
Hayden, Colo	15.3	15.1	31.8	48.9	4.0	10,680	10,570
Renton, Wash	14.7	15.8	33.2	40.5	11.6	9,868	10,530
Roundup, Mont	12.3	17.7	31.9	47.4	8.4	10,890	11,320
Boulder, Colo	19.3	19.4	31.8	43.5	5.4	9,781	9,820
North Park, Colo	21.0	20.6	33.4	42.3	3.3	9,710	9,540
Sheridan, Wyo	23.6	23.6	32.6	38.9	4.9	9,175	9,170
Colestrip, Mont	24.3	24.9	28.0	40.5	7.2	9,230	9,450
Gillette, Wyo	30.8	30.8	30.3	33.9	5.0	8,120	8,120

The solution to this difficulty appears to be to draw the line lower in the group, say at a moisture content of 13.5 per cent., and consider the Hanna, Mount Harris and Gallup coals as belonging in the same class as the Superior coal of Wyoming. These coals, while slacking to a certain extent on exposure to the weather, do not slack so readily or to such an extent as to detract greatly from their excellence and they will bear fairly well transportation in open cars. The lower limit of this group is also uncertain, as shown by the last analysis in Table 2. This sample is from the thick bed of coal recently opened at Gillette, Wyo., which was classed by the collector as subbituminous. The writer has not seen this coal and therefore can add nothing to what has been reported from the chemical laboratory. It raises the question as to whether a moisture content of 27 per cent. or one of 30 per cent. should constitute the dividing line between the browncoal and hydrobituminous groups. This question should not be decided until a careful field study is made of the coals in the border zone.

The writer is of the opinion that few, if any, of the lines of separation between major groups or minor classes can satisfactorily be made without exhaustive field studies, except in cases of coals that are well developed,

and have been on the market for a number of years, so that their physical and chemical properties have been thoroughly tested, and even then it may be necessary to give particular attention to some phase of the problem that apparently has been neglected. In the present case the limits set by the writer are provisional only and are given merely for the sake of focusing attention on them and drawing the fire of those who have a different opinion and are familiar with the facts in the case. On this basis the writer would define Class A of the hydrobituminous group as extending from a moisture content of 27 per cent. to one of 13.5 per cent.

The coals of this class are susceptible of being divided into at least two types—xyloid coals and cannel coals—but under present conditions the recognition of these types is not an important matter.

Class B—Low-moisture Coals

The coals of this class are better in many ways than those of Class A, because they contain less moisture, slack less or not at all on exposure, and are generally harder and bear long transportation without serious degradation. In a general way this class of coal may be said to include most of the better coals of the Mississippi Valley. The upper limit is arbitrarily placed but should be set so as to separate the coals of this field from those of the Appalachians. This line, wherever it is placed, will cut through some important coal field that is a large producer at the present time and the operator who is assigned to a lower class will not take that assignment cheerfully. We should, after deciding on the approximate position of the line, consider the matter carefully to see if a slight shift in position may not more nearly fit the field conditions, and thus avoid laying ourselves liable to the charge that we disregarded the facts and so have a classification that is not usable and will not be generally accepted. With this in mind, the writer would suggest that in most cases we first establish a twilight zone between the various classes and then endeavor to find which will best fit the facts in the field and cause least opposition. The writer would suggest that the dividing line between Class B of the hydrobituminous group and Class A of the bituminous group be drawn at some point between 5 and 6 per cent. of moisture.

It is altogether probable that the two types of coal, the xyloid and the cannel types, will be found in this class. The xyloid coals are very abundant but cannel coals are seldom found. The writer does not recall having seen a cannel coal in this class nor to have seen an analysis of such a coal, but doubtless they are present as they are found in all coals up to and including most of the bituminous group—the next higher group in the series.

Bituminous Group

The bituminous group as recognized by the writer in this paper is very similar to the group now generally called by that name. The upper part is exactly the same but a part of the base of the group, as heretofore recognized, is separated and placed in the hydrobituminous group. The bituminous group may therefore be described as having at its base coals whose moisture content is about 5 or 6 per cent. and extending upward in rank to and including the low-volatile or so-called "smokeless coals," as at present understood. The coals of this group are decidedly the most valuable coals in this country, as they include all of the coking coals, the best gas-making coals, and in the uppermost class, the coals that rank with the best steaming coals of the world.

The basis for classification in this part of the series is very different from that which has been used in the groups and classes in the coals of lower rank. Moisture, which has been such an important factor in the latter coals, has ceased to play a part in the classification of these coals, as its variation is slight in all parts of the bituminous group; it is in all cases less than 6 per cent. and in some cases falls considerably below this figure. The only factors shown in the proximate analysis that remain are volatile matter and fixed carbon and these seem, in a general way, to be fairly satisfactory for classification purposes. The change from lower to higher rank is one that is to be expected, for the effect of rock metamorphism is first to drive off most of the water contained in the plant tissues and then, as the heat or pressure increases, to break up the hydrocarbons of the vegetal material and to drive off the more volatile portions. As a result of this process the volatile matter steadily decreases as the coals increase in rank, and conversely the fixed carbon, which is the part least affected, shows a steady increase in proportion to the total mass. The decrease of the volatile matter or the increase of the fixed carbon affords a reasonably accurate measure of the metamorphism of the coal.

How shall these differences be expressed and what kind of an analysis shall we use? The ultimate analysis is undoubtedly the most satisfactory one, as it gives the elements which make up the coal rather than a group of compounds which may never be twice the same. But while the ultimate analysis is undoubtedly the most accurate, it is certainly never used by the practical man to express the quality of his coal. In fact, it is doubtful if one operator or dealer in one hundred ever used an analysis of this kind or knows what it means. In addition to this, it must be remembered that it costs much less to make a proximate than an ultimate and that by far the greater number of analyses that have been made and are available for our use are of the proximate variety. Because of all of these facts there seems to be no question regarding the superiority

of the proximate over the ultimate analysis for classification purposes where the classification is to be applied to ordinary affairs of life and used by the ordinary layman.

While the proximate analysis is the one to be used, it must be applied in a proper manner or the results will be conflicting. The writer has never heard an operator or dealer refer to the rank of his coal other than to say it has such and such a percentage of volatile matter, without specifying whether this is the percentage of the whole content of the coal or only of a limited portion. The proximate analysis contains two variables, moisture and ash; and to specify the percentage of volatile matter without at the same time saying whether or not the moisture and ash have been eliminated is confusing, to say the least.

In order to avoid this difficulty, Persifor Frazer, of the Second Geological Survey of Pennsylvania, proposed to use the "fuel ratio," or the quotient of the fixed carbon divided by the volatile matter, instead of giving the actual percentages of either. This has the great advantage that it can be obtained in a moment's time from any proximate analysis whereas to compute the percentage of the fixed carbon on a moisture-free and ash-free basis requires a rather lengthy computation. The only objection to Frazer's method has been that in the lower part of the bituminous group of coals the fuel ratio becomes a very small fraction. This however does not appear to be of sufficient weight to cast it aside and the writer strongly urges that fuel ratios be used in classifying most of the high-rank coals.

Two distinct classes of the bituminous group are universally recognized in this country. These are Class A, high-volatile coals, and Class B, low-volatile coals. These classes have decided characteristics, both chemical and physical, and generally one can tell at a glance to which class a given coal belongs. The division of the group into three classes has been strongly urged but there does not seem to be unanimity of opinion in favor of this proposal. According to those who favor this plan, the bituminous group should be divided into three classes, as follows: Class A, high-volatile coal; Class B, medium-volatile coal; and Class C, low-volatile coal. The writer does not favor this plan of subdivision, for the following reasons: (1) The number of divisions, whether of major or minor orders, should be kept as low as possible, because each new one adds to the complexity of the scheme and to the practical difficulty of making the division in the field; (2) The distinction between a high-volatile coal and one of medium volatile is based largely upon use, only slightly upon chemical composition, and not at all on physical characteristics. The writer feels that the threefold division of the bituminous group should be very carefully considered before it is adopted or rejected, but from his own point of view, it belongs to a use classification rather than a general classification. In the present treatise, therefore, it will be omitted.

Class A—High-volatile Coal

This class includes all of the lower part of the bituminous group, consequently its lower limit coincides with the lower limit of the group, or at the point where the moisture content is about 6 per cent. The upper limit is determined by its fuel ratio and is therefore a function of the fuel content of the coal. From the base of the group upward the volatile matter decreases steadily and the fixed carbon increases until the limit of Class B is reached. As the limits of Class B, the low-volatile coals, have been much more definitely determined than have those of Class A, the former will be used in fixing the dividing line between them. The writer is convinced that the fuel ratio of the lowest low-volatile coal should be fixed at 2.5. This limit agrees closely with the limit fixed by the trade and, so far as the writer is aware, will not cause an undue amount of friction, even in case it should be legally enforced. If this line is agreed to, the upper limit of Class A may be described as falling just below the fuel ratio of 2.5.

The coals of Class A are so well known that it does not seem necessary to describe them further than to say that in general they are hard and show little degradation on handling and in shipment. The different types of coal are well developed in this class, particularly the cannel coals as contrasted with the more common xyloid coals. It is possible that other types may be recognized as more detailed studies are made but at the present time these are the two most commonly recognized, and it does not seem wise to attempt to introduce others until they have been shown to have very distinct properties and physical characteristics.

Class B—Low-volatile Coal

This is one of the most valuable kinds of coal, and is largely restricted to the Appalachian fields. It is characterized by its friableness, which apparently is caused by the great earth pressure to which it has been subjected. Most of the fields of this class of coal are well developed and many data are available regarding its chemical composition. According to the records of the U. S. Bureau of Mines and the U. S. Geological Survey, the highest rank coal of this kind is found a short distance below Welch, West Va., where the coal has a maximum fuel ratio of 5.37. As it is possible that fuel ratios slightly higher than this may be found locally in some of the fields of this kind of coal, the writer considers that the upper limit of Class B should be placed at a fuel ratio of about 5.5. As this fuel ratio is greater than that of some of the coals of Montgomery and Pulaski counties, Virginia, which, according to the writer, belong in the next higher group of coals, it appears necessary not only to

define Class B or low-volatile coals as having a certain range in their fuel ratios, but also as being extremely friable, as this characteristic will differentiate them decidedly from the harder coals of the anthracite group.

In the low-volatile class the writer has frequently found coal having the physical characteristics of cannel coal, but the chemical properties are exactly the same as the adjacent normal low-volatile coal. As the peculiar property that gives to cannel coal its value as a grate fuel is its high percentage of volatile matter, it necessarily loses this value when its content of volatile matter becomes less than that of its fixed carbon, despite the fact that it still retains the physical characteristics of a typical cannel coal. These physical features prevail into the anthracite stage of metamorphism but no one would think of an anthracite as being a cannel coal. In view of this statement, the writer contends that in the low-volatile class only one type of coal should be recognized.

Anthracite Group

The friable coals of the low-volatile class, under increased pressure, gave off some of the volatile matter and at the same time suffered recementation which results in a very decided increase in hardness, producing coals that are generally referred to as anthracites. These coals are of varying degrees of hardness and contain so little volatile matter that they generally burn with a blue flame and produce little smoke. They are essentially a domestic fuel, and as such are of great importance. Most persons when they hear the term "anthracite" mentioned visualize at once the hard dry anthracite of Pennsylvania, but the anthracite group contains several kinds of coal, all of which are characterized by a small content of volatile matter, a large content of fixed carbon, and generally much greater hardness than that of any coal lower in the series.

Although anthracites are well known, there is a surprising lack of reliable data regarding their chemical composition and their physical properties. Much of this lack is due to the fact that fields of this kind of coal are generally small and rather widely separated and it is difficult to find the place where one kind of anthracite merges into another kind, either higher or lower in rank. Because of this condition it is impossible at present to set definite limits on the subdivisions of the group that in a general way can and should be made. The anthracite group is divided into three classes, as follows: Class A, or high-volatile anthracite; Class B, or low-volatile anthracite; and Class C, or meta-anthracite. The coals in Class A have heretofore been called semianthracite; the coals in Class B, hard dry Pennsylvania anthracite; and the coals of Class C, Rhode Island anthracite, graphitic anthracite or superanthracite.

Class A—High-volatile Anthracite

The term "semianthracite" heretofore applied to coals of this class is very objectionable, as it implies that the coal is only half as good as the typical anthracite of Pennsylvania whereas its heating value may be considerably greater. As the term has given much trouble in the past, the writer would strongly recommend that it be dropped. The principal difference between this coal and typical anthracite is that the coal of class A is not so hard and it carries a greater percentage of volatile matter, so that it ignites more readily and for 15 or 20 min. burns with a short yellow flame until the volatile matter is largely consumed; after this, it burns with a blue flame like typical anthracite.

The typical coal belonging to this class in Lykens Valley No. 5 bed in Dauphin County, Pa. This is a small basin lying west of and connected with the main basin of the Southern Anthracite field. Similar coal is mined in outlying basins in Sullivan and Tioga counties, Pennsylvania; in the Valley fields in Montgomery and Pulaski counties, Virginia; and in and about Spadra and Russellville, Arkansas. At most of these places the basins containing anthracite are widely separated from those containing low-volatile coal of the next lower rank; in Arkansas, however, the two merge, but there are few data available regarding the character of this transformation.

The chemical composition of the coals of Class A of the anthracite group, together with two representing the adjacent coals above and below, is shown in Table 3.

TABLE 3.—*Analyses of Coals of Class A, Anthracite Group*

	Fuel Ratio	Moisture	Volatile Matter	Fixed Carbon	Ash	B.t.u.
McCoy, Va.	4.7	1.6	12.8	60.7	24.9	11,190
Blacksburg, Va.	4.9	1.9	14.0	68.9	15.2	12,740
Parrott, Va.	5.3	1.9	11.8	61.7	24.6	11,200
Capels, W. Va.*	5.4	2.2	14.5	78.1	5.2	14,470
Clarksville, Ark.	5.4	2.7	13.7	74.5	9.1	13,480
Blacksburg, Va.	5.8	2.7	11.9	68.4	17.0	12,390
Russellville, Ark.	6.3	2.8	11.9	75.2	10.1	13,360
Empire, Va.	6.8	2.1	10.2	68.7	19.0	12,110
Merrimac, Va.	7.3	2.1	9.7	71.1	17.1	12,330
Pulaski, Va.	7.8	4.5	8.2	63.9	23.4	10,880
Russellville, Ark.	8.0	2.1	9.8	78.8	9.3	13,700
Lykens, Pa.	8.3	1.1	9.6	79.5	9.8	13,590
Merrimac, Va.	8.5	2.5	8.8	74.6	14.1	12,880
Bernice, Pa.	8.9	3.3	8.4	72.8	15.5	12,500
Carbondale, Pa.*	9.6	2.4	6.6	63.3	27.7	10,310

* These coals do not belong to Class A, but are inserted for comparison with the coals of the overlying and underlying ranks.

The coal from Caples, West Va., is the highest rank low-volatile coal of which the Geological Survey has any record, and the coal from Carbondale, Va., is generally regarded as about the lowest rank coal of Class B of the anthracite group. The Caples coal is very interesting, as its fuel ratio is higher than the lowest, or McCoy coal Montgomery County, Virginia, although the latter is a fairly hard coal and sells on the market as a kind of anthracite. The Caples coal, on the contrary, is an exceedingly friable coal and under no circumstance would be considered as an anthracite, as it is a typical Pocahontas coal and not well suited to domestic use, unless it were converted into coke.

The writer in reporting on the Valley coal fields to the State Geologist of Virginia (*Bull.* 25. Va. Geol. Survey), was greatly puzzled as to the best manner of dealing with this problem. At first he took the position that the lower limit of Class A of the anthracite group should be placed at a fuel ratio of 5, but here is a field in which the fuel ratios range from 8.5 on the southeast side of the field (the side from which came the great thrust that metamorphosed the coal) to 4.7 or possibly lower on the northwest side, and a line of separation drawn at the point where the fuel ratio is just 5 would split a small field directly in two, despite the fact that coals from both sides of this line are being sold on the market as "Virginia anthracite," and the writer can see no reason why they should not continue under some such name to serve this market as a domestic fuel. The mere fact that the definition says that the lower limit of the high-volatile anthracites should be placed at the fuel ratio of 5 does not meet the needs of the case and the physical properties of the coal should be considered in fixing this limit; conversely, the fuel ratio of coals in Class B of the bituminous group can not be limited by a fuel ratio of 5, but should include all friable coals, no matter whether they fall below or above that limit. The mere classing of the Caples coal as an anthracite will not make it so, because it lacks the physical properties of anthracite, and these are all important.

The conclusion of the writer is that coals of Class A of the anthracite group should be so defined that both the chemical and physical properties will be taken into account and that in case the former seems to fail the latter should decide the case.

The upper limit of Class A of the anthracite group can not be definitely fixed because of lack of reliable analyses of coals in the debatable zone between the clearly recognized members of Classes A and B. Table 3 affords some evidence on the subject, as the coal at Carbondale, Pa., is generally considered as belonging to Class B, the lowest member of the hard dry anthracites. If that is accepted, the fuel ratio of about 9.5 should be regarded as the line of separation between them. This, however, needs further confirmation.

Class B—Low-volatile Anthracite

This class is intended to include what is generally regarded as typical anthracite or the hard dry anthracites of the Pennsylvania fields. The writer has little evidence in the form of chemical analyses by which to fix the limits or describe the character of the coals of this group. The lowest coal, so far as the data at hand indicate, is the Carbondale coal listed on a previous page with a fuel ratio of 9.6, and the highest is one from St. Nicholas with a fuel ratio of 76.0. There is, therefore, a wide range in the composition of Pennsylvania anthracite and with the meager data at hand it is impossible to fix a definite upper limit.

Hardness and high fixed carbon content are the characteristics of the anthracites of Class B and these give to it a particularly high value as a domestic fuel. The structure of the coal is not entirely homogeneous and here and there traces of cannel-coal structure are still visible, but the distinguishing feature of a cannel coal, its high volatile content, is entirely absent.

Class C—Meta-anthracite

It is rather difficult to visualize a stage in the metamorphism of coal that lies above and beyond that of anthracite, but here and there such coals are known. The coals that are obviously above the anthracites in rank are of different compositions and all that can be said about them at the present time is that they are beyond the stage of anthracization, hence the name "meta-anthracite" has been chosen for them.

The largest and best known area of this kind of coal is the field lying in Rhode Island and Massachusetts. This coal is frequently referred to as graphitic anthracite. It has clearly passed beyond the stage of a straight anthracite and has the peculiar characteristic of a moisture content of from 13 to 14 per cent. This is not extraneous moisture, as has been proved by McFarlane, who found the moisture content of a coal in direct contact with a large basalt sill to be 11.1 per cent., though the same coal at a greater distance from the sill is a normal low-moisture anthracite.[2]

Another type of meta-anthracite was found by the writer near Poolsville, Md. which is supposed to have been altered by proximity to a large igneous dike. This coal is without moisture and with a hardness of about that of steel. So little is known about these coals that it is not wise at the present time to attempt to describe them, but merely to recognize them as constituting a class which has been metamorphosed until they have passed the stage of ordinary anthracites and consequently are properly referred to as meta-anthracites.

---

[2] G. C. McFarlane: Igneous Metamorphism of Coal Beds. *Econ. Geol.* (1929) **24**, 1.

Summary of Groups and Classes

The proposed grouping of the coals and the names to be applied to the various groups and classes are best shown in the following outline:

Coal	Browncoal group.......	A. Peat class
		B. German brown coal
		C. Lignite class
	Hydrobituminous group	A. High-moisture class
		B. Low-moisture class
	Bituminous group......	A. High-volatile class
		B. Low-volatile class
	Anthracite group.......	A. High-volatile class
		B. Low-volatile class
		C. Meta-anthracite class

DISCUSSION

G. H. Cady, Urbana, Ill. (written discussion).—The classification proposed by the author is of peculiar interest to the State Geological Survey of Illinois because it introduces a new group of coals in which it is proposed to place practically all, if not all, the coals mined in Illinois. The criterion by which this particular group of coals—the hydrobituminous group—is identified is the "as-received" moisture content, the lower limit of the group being 25 per cent. moisture content, and the upper limit 5 to 6 per cent. It is further proposed to subdivide the group into two classes, one class being represented by coals having a moisture content above 13.5 per cent., and the other class by coals having less than 13.5 per cent. moisture within the limits of the group. This proposed hydrobituminous group of coals includes essentially what has been commonly called the Low Rank Bituminous coal. Although in some particulars the new name is an improvement in nomenclature it is unfortunate in emphasizing the moisture content in coals having such a wide range in the amount of moisture present. It seems very doubtful whether the name would be regarded as a happy selection by those interested in the commercial exploitation of Illinois coal. So far as Illinois coals are concerned there is some evidence that the group, aside from the name, is valid. Whether there are outside of Illinois coals of low rank and also low moisture content or of high rank and high moisture content would determine the general validity of the group.

Even if the group be recognized, differences of opinion are likely to arise in regard to the advisability of subdivision or in regard to the number and limitations of the subdivision proposed. The nature of the differences in the moisture content of Illinois coals and the definiteness with which discrimination is possible between different coals on the basis of moisture content is a matter of importance in determining the advisability of any subdivision, the validity of the subdivisions proposed, or the necessity of the revision of the proposals.

The hundreds of carefully collected face samples made by the State Geological Survey show in their analyses, made by the U. S. Bureau of Mines and by the Testing Laboratory of the University of Illinois, fairly definite facts in regard to the distribution of variations in moisture, ash, volatile matter, fixed carbon, sulfur, and calorific character. County average[3] values that have been prepared by the State Geological

[3] A. Bement: Illinois Coal. Illinois State Geol. Survey *Bull.* 56 (1929) 99.

Survey for presentation in a recently published bulletin are thought to be closely representative of the coals mined in the respective counties, and they provide an excellent basis for generalizations in regard to characteristic variations.

The item of the proximate analysis that shows the greatest variation in the face samples of Illinois coal is the moisture. The general nature of the variation in moisture

Fig. 1.—Average moisture content of coal beds of Illinois by county and by seam.

for the area of the Illinois coal basin is shown in Fig. 1, upon which is entered the average moisture content of the coals by county and by seam. That there has been progressive desiccation of the coals southward is apparent; but it will doubtless be observed that this decrease in moisture content is not regular. The decrease across a narrow zone in the extreme southern part of the coal basin which includes Randolph,

Perry, Franklin, White, Jackson, Williamson, Saline and Gallatin counties is of approximately the same amount as the decrease from the northern part of the coal basin in La Salle County down to the north edge of this zone. In the zone itself there is a general decrease from north to south in the moisture content of the coal, but the direction of greatest decrease is toward the southeast, that is toward the eastern boundary of Gallatin County.

Fig. 1 and Fig. 2, which shows the variations in moisture content by isohumes, show that the area in the state wherein the coals have a moisture content of less than

FIG. 2.—ISOHUME MAP OF ILLINOIS, SLIGHTLY GENERALIZED; IRRESPECTIVE OF SEAM.

10 per cent. is relatively small, and that variations in this area are fairly regular and progressively smaller toward the southeast, there being successive relatively narrow belts in each of which the range of variation is slight.

In establishing the definition of the classes within the hydrobituminous group it appears that only empirical criteria have been and as yet can be used. That is to say, there is no naturalistic basis for preferring one point of division above another, since, so far as the writer is aware, it has not been suggested and much less demonstrated that decrease in the moisture content of these high-moisture coals is at any

step accompanied by critical changes in the immediate constitution or character of the coal substance. It follows, therefore, that classification must necessarily be based upon empirical considerations, at least until a naturalistic basis can be found. The practical usefulness of 13.5 per cent. moisture as a basis of separating the group into classes as compared with other boundaries that might have been selected therefore merits consideration. Since no boundary can be selected which will definitely place all coals as of one or the other class, the boundary should be so placed that the area underlain by coals occupying the transitional position between the two classes shall be as small as possible. It would also be well if classes were representative of differences recognized commercially. Since, as was pointed out in the preceding paragraph, variations within short distances are greater in southern Illinois than elsewhere in the state, the area underlain by coal having a narrow range of moisture content will be smaller in this part of the state than elsewhere. It appears advisable therefore to place the boundary between classes within the limits of the range existing in southern Illinois.

In the selection of percentage to use as a basis for classification, such that the transitional zone will be geographically narrow, there is, on this basis alone, little choice with respect to any amount between 5 and 10 per cent. of moisture. Obviously, however, since the upper boundary of the group is at about 5 per cent. the lower boundary of the upper class must be far enough below the upper boundary to provide the inclusion of a reasonable extent of coal. For this reason, it would be well to place the boundary between classes as low, that is, numerically as high as possible still keeping the geographical extent of the transitional area narrow. This condition seems to be fulfilled, so far as Illinois coals are concerned, at about 9.5 to 10 per cent. The transitional or border zone is relatively narrow running through Randolph, Perry, Washington and Marion counties and fairly definitely separates the lower moisture coals of Jackson, Franklin, Williamson, Saline and Gallatin counties from the higher moisture coals of St. Clair, Clinton and Bond counties and counties to the north. The distinctions are those, likewise, that are recognized commercially, particularly that one between the coals of the northern and the coals of the southern counties. The intermediate character of the coal in the intermediate zone in Randolph, Perry, Washington (eastern) and Marion counties is also commonly realized commercially.

So far as Illinois coals are concerned there appear to be some reasons for regarding as valid a distinction based upon a moisture content of more than or of less than 9.5 to 10 per cent. By almost any system of classification—B.t.u. values, unit coal values, carbon ratio, and various graphic schemes—the southern Illinois coals having a moisture content of less than 9.5 to 10 per cent. are definitely distinguishable from other coals in the state, and by all such classification schemes a narrow transitional or intermediate group of coal, mainly those mined in Perry County, is recognizable.

The writer has not given critical consideration to the value of a subdivision of the group into two classes, one above and one below 9.5 to 10 per cent. moisture, in other coal fields than Illinois. A hasty inspection of tables of analyses shows that most of the coals mined in Western Kentucky would fall into the class having less than 10 per cent. moisture. In fact, the class would apparently include the coals of southeastern Illinois, western Kentucky, and possibly the southern part of the Indiana field in Knox, Gibson, Pike and Warrick counties, although the coals of Indiana lie very close to the boundary between the two classes. It is probable that better discrimination would be possible by placing the boundary at 9.5 per cent. than at 10 per cent. So far as Illinois is concerned it makes little difference whether the boundary is placed at 9.5 or 10 per cent.

The mathematically unequal subdivision of the hydrobituminous group which would result from the adoption of a proposal to make the place of separation 9.5 per

cent. moisture raises the question whether or not the class which extends over the larger range in moisture values, that is, from 9.5 to 25 per cent. might not be divisible into two or more classes. It is, of course, possible to make a subdivision on an empirical basis at any stated moisture content, but presumably a position would be chosen, if at all, because it appeared to accord with natural distribution of values. Inspection of tables of analyses and maps giving the distribution of moisture values shows that there is a fairly definite geographic limitation in the distribution of coals having more than 14 per cent. moisture and those having 9.5 to 14 per cent. The coals of central and southwestern Illinois, that is, those mined near Danville, Springfield, in the Standard district of Macoupin and Montgomery counties, and in the Belleville district have the lower moisture content, and those coals mined in areas to the north in the Peoria, Rock Island, La Salle and Wilmington regions have the higher moisture content. Analyses[4] of Indiana coals published by the Bureau of Mines rarely show more than 14 per cent. moisture. Coals from the northern and central part of the Indiana coal field on the basis of moisture content would be classified with coals from the central and southwestern part of the Illinois field. Fourteen per cent. is sufficiently near to the 13.5 per cent. suggested by Dr. Campbell so that they can be considered the same for the purposes of classification. The advisability of establishing a boundary between classes at this point is, however, doubtful, so far as Illinois coals are concerned, because the geographical area underlain by coals transitional between the two adjacent classes is broad and poorly defined, and the classes would not be substantiated by commercial distinctions. Variations in moisture content in successive samples of coal from the same mine are considerably greater among high-moisture coals than among low-moisture coals and it would be very difficult to fix definitely the classification of a coal with a moisture content within a per cent. or two of the limiting value. It seems probable that the usefulness of a subdivision of the group at some point higher than 10 per cent. depends upon the distribution of moisture content in other states than Illinois and Indiana.

Before making a concluding statement in regard to the advisability of subdivision or in regard to the nature of the subdivision if made it would be well to consider the specific effect of subdivision into two classes upon certain of our coals.

The number of shipping mines operated in 1928 was 206. There would be very little doubt in regard to the class of coal into which the product of the individual mines would fall, using 9.5 per cent. of moisture as the division between classes, for at least 90 per cent. of these mines. The remaining 10 per cent. lie in the transitional area in Washington, Perry, Randolph, southeastern St. Clair and southwestern Marion counties, in an area which is likely to become of immediate increasing and soon of great importance because of the large areas of strippable coal within it. The writer is somewhat at a loss to know just how the classification problem would be met in this region other than by assigning the coal to a transitional position between two classes, which would simply mean the erection of a third and very narrow class. If this difficulty would arise in a region relatively small in geographic extent the difficulty would appear to be still greater if a boundary is selected such that the transitional zone is represented by coals which extend beneath a large area.

There are in addition to these transitional coals certain other coals of a more or less anomalous character the classification of which will be sure to make trouble for any classification based like this one upon empirical criteria. For example, we have in Will County an area of No. 2 coal for which Bureau of Mines analyses of face samples show 14.4 to 16.2 per cent. moisture, a carbon ratio of 56.6 and 56.4, and a pure coal B.t.u. value of 14,280 and 14,300 units, respectively. The carbon ratio is essentially the same as that of No. 6 coal in the east part of Perry County where the

---

[4] Analysis of Indiana Coals. U. S. Bur. Mines *Tech. Paper* 417 (1927).

moisture content is about 9.5 per cent. and the B.t.u. value is greater than that of eastern Perry County. It appears to be only in the relatively high content of moisture that this coal differs from coals in southern Illinois having a moisture content less than 9.5 per cent. It would possibly be wrong to classify this coal with others in northern Illinois having similar moisture content but quite different carbon ratio. For example No. 2 coal in Grundy County lying west of Will County shows an average moisture content of 16.8 per cent., a carbon ratio of only 50.8 and a "pure coal" value of 14,280 B.t.u. It is not usual to find coals with a carbon ratio exceeding 50 in the Longwall District of northern Illinois which includes Bureau, Putnam, Marshall, La Salle and Grundy counties. At another mine, however, in the southern part of the Longwall District in Woodford County the carbon ratio is 57 to 58 with a moisture content of about 15 per cent. These anomalous relationships suggest that a classification based upon the stage of coalification as indicated by the moisture content of a coal fails to take proper account of the effect of coalification upon the relative amount of fixed carbon and volatile matter.

There is the further anomalous case of the No. 6 coal at Lovington, Moultrie County, in a position nearly 100 miles north of the main body of low-moisture coals in southern Illinois. The moisture content of this coal is 6.7 per cent., the carbon ratio only 52.2. On the basis of moisture content this coal should be grouped with those of southern Illinois, although in doing so there appears to be a violation of the general rule that carbon ratio is an index of the amount of coalification.

Instances of anomalies of this kind are not numerous, but the more we learn of the chemical character of our coals the less certain we are that differentiation based upon a specified quantity of any one of the substances itemized in the proximate analysis will result in a classification various items of which can be arranged in order to represent successive stages of coalification.

In conclusion, the opinion is expressed that although the group described may be a valid one the name selected is not desirable although possibly preferable to the common name Low Rank Bituminous. If subdivision of the group is made, it is believed that so far as Illinois coals are concerned it is more satisfactorily accomplished at 9.5 per cent. of moisture than at a higher level up to at least 18 per cent., which is somewhat above the mean maximum moisture content of Illinois coal. Basis for subdivision at a level above 18 per cent. would have to be found in other states. So far as Illinois coals are concerned it is believed that subdivision into classes is impractical, and possesses no advantages beyond those offered by a simple statement of moisture content. Presumably the classification should mean something in terms of the coalification process and an empirical subdivision carries no such significance. It is also impossible to make a subdivision into classes that will not leave a fairly large number of important coals in an intermediate position, the relative positions of which will after all be defined in terms of moisture content. The range of the moisture content in Dr. Campbell's hydrobituminous group of coals is sufficiently large so that a mere statement of moisture content of a coal will thereby satisfactorily establish its relative position, if it can be accepted as established that in this group rank or stage of coalification is definitely determined by moisture content. It seems quite possible that additional evidence of the truth of this premise is necessary.

R. J. Holden, Blacksburg, Va. (written discussion).—Various classifications of coal have been proposed and most of these have been based on chemical composition. There seems to be some objection to all of these proposed classifications and each succeeding scheme seems to have been an attempt to obviate some of the failings of previous classifications. When the Frazer classification,[5] using fuel ratios, came out

[5] P. Frazer, Jr.: Classification of Coals. *Trans.* A. I. M. E. (1877–78) **6**, 430.

it served its purpose as applied to Pennsylvania coals, but when western and softer coals came into use the Frazer plan failed to differentiate the softer coals from each other. Various new classifications were proposed with their chief purpose to separate the softer coals. The Campbell plan,[6] using the carbon-hydrogen ratio, satisfactorily separates the middle ranks but requires an ultimate analysis which is not always available nor is it easily made. None of the various other plans using proximate analyses, such as Parr's,[7] Collier's,[8] Grout's,[9] Dowling's[10] and others, have met general approval. Fuel ratios have been most satisfactory for middle and higher ranking coals but there is some question as to what figures should be limiting values. In 1925 Campbell[11] called attention to the fact that modern approved methods of making proximate analyses yield higher volatile matter than former methods and proposed a change in some of the limiting ratios given by Frazer. He also departs from the prevailing plan of classifying coals exclusively on chemical composition and says that for coals of the higher ranks hardness and color of flame are criteria.

In his introduction to the present discussion of classification of coal Fieldner calls attention to the fact that the Technical Committee on Scientific Classification was requested to formulate a system of coal classification based on chemical and physical properties of the coal. The striking feature here is the support of the idea just given, namely, that we depart from the former plan of using only chemical properties and use physical properties as well. In the oral presentation of the papers something has been said about physical properties as a basis of classification. In the papers there are also references to the same plan. Campbell says that previous classifications have failed because they did not recognize that rock metamorphism is the cause of most of the characteristic differences of coal and, therefore, this should be a prime factor in its classification. He does not here specifically refer to the physical effects of metamorphism on the coal but in his subsequent discussion of classes he describes these.

In the physical effects of metamorphism there is some analogy between the metamorphic changes in coals and other sedimentary rocks. In the case of unconsolidated sands there is metamorphism of the sand by cementation to a sandstone and of the sandstone by further cementation to quartzite. When the quartzite is subjected to suitable pressures it is fragmented by granulation but with increasing pressure there is a minimum size of particle and further pressure produces not finer granulation but recrystallization and the rock becomes harder instead of more friable. In a similar manner with pressure peat hardens to lignite and lignite to bituminous coal. When the coal has reached a stage of metamorphism in which the fixed carbon has reached 70 per cent., it is apt to be much more broken than coal with a lower percentage of fixed carbon. With an increase of fixed carbon to 85 per cent. the coal is apt to be materially harder and with a higher percentage of fixed carbon still harder. If these hardnesses could be expressed mathematically and the values plotted against fuel ratios on rectangular coordinates, the result would not be a straight line but a curve. In general this curve would have low points for peat, high points for bitu-

---

[6] M. R. Campbell: The Classification of Coals. *Trans.* A. I. M. E. (1906) **36,** 324.

[7] S. W. Parr: Composition and Character of Illinois Coals. Ill. State Geol. Survey *Bull.* (1906) 3.

[8] A. J. Collier: The Coal Resources of The Yukon, Alaska. U. S. Geol. Survey *Bull.* (1903) 218.

[9] F. F. Grout: The Composition of Coals. *Econ. Geol.* (1907) **2,** 225.

[10] D. B. Dowling: Classification of Coal. Can. Min. Inst. *Bull.* (1908) No. 1, 61.

[11] M. R. Campbell: The Valley Coal Fields of Virginia. Va. Geol. Survey *Bull.* 25, (1925) 125.

minous, lower points for semibituminous, high for semianthracite, and still higher for anthracite.

Hardness is a general term and may include various phases as resistance to abrasion, friability, resistance to impact, crushing strength and other properties, and a mathematical expression for one of these phases might not agree with those for some other phases.

Friability has received the most attention.[12] Smith used a drop test on sized lumps, then sized the product on screens of different sizes. He seemed chiefly concerned with obtaining a mathematical value for what he calls the degradation. He does not give the chemical compositions of the coals tested and therefore it is not possible to get from his work a definite relation between the chemical and physical effects of metamorphism, but the composition can be inferred roughly from the kinds of coals which he used. He obtained the following degradation figures: cannel coal, 7–10; Illinois bituminous, 22–32; Pocahontas, 39; anthracite, 12. The significant figures here are the high values for the Pocahontas, the low figures for the anthracite and the medium figures for bituminous.

Nicolls used a rotating cylinder to produce the breakage and sized the product on screens of various sizes. He expressed his results in percentages passing the several screens. He has made proximate analyses of his samples but he did not attempt to produce a numerical value for friability. Considering the fact that he showed four different percentage products for the four different sieves, without reducing these to single numerical value it is not possible to give any exact figures of friability but roughly, by inspection, his results show the highest values for the Pennsylvania anthracite, the lowest values for Pennsylvania semibituminous, high values for bituminous and higher values for lignite. This is roughly in keeping with Smith's results and both of them show the outstanding fact which is in agreement with the statement previously made and with results obtained commercially in shipping sizes: that the semibituminous coals are the most friable; that the bituminous coals are less friable, and that the most metamorphic and little metamorphic coals have the lowest friability.

If the percentage compostions obtained by proximate analyses on coals of different ranks be reduced to ash-free, moisture-free basis and the values for volatile matter and fixed carbon plotted on rectangular coordinates, the result will be a straight line. As friability seems to plot a U-shaped curve and the chemical properties a straight line, it appears that judging from one physical property the chemical results of metamorphism do not parallel the physical results of metamorphism. Such being the case, the use of physical properties introduces new criteria for differentiating coals of different ranks.

---

[12] C. M. Smith: An Investigation of the Friability of Different Coals. Univ. Ill. Eng. Exper. Sta. *Bull.* (1929) 196.

J. H. H. Nichols: Friability Tests on Various Fuels Sold in Canada. Canada Dept. of Mines, Mines Branch, *Investigations of Fuels and Fuel Testing* (1924) 644, 20.

Coal Classification; a Review and Forecast

By George H. Ashley,* Harrisburg, Pa.

(New York Meeting, February, 1930)

At the beginning of the war, about 13 years ago, a conference was called in Washington to lay plans for pooling the coals of the United States. A careful review of the various systems of classification then extant showed none well adapted to the needs of coal pooling. The system of Tidewater pools then adopted, using numbers to distinguish one pool from another, was based on typical coals of different localities.

At that time, the writer undertook the preparation of such a practical classification. Aside from drawing on his experience of over 20 years in the coal fields of the United States, reaching from the Pacific Coast and the Rio Grande to Rhode Island, he first examined and carded all of the large collection of coals in glass jars made by and for M. R. Campbell. Next, he carded all of the descriptions of coals he could find in the U. S. Geological Survey bulletins and elsewhere, as well as practical tests of coals by the Navy and other organizations and then took account of every published coal analysis of the U. S. Geological Survey and U. S. Bureau of Mines. A study of this information, involving thousands of recomputations, led to the presentation of a paper on "A Use Classification of Coal" before this Institute at the Chicago meeting, September, 1919.

That classification was unique (1) in a classification based on "standard coal" which included moisture, but used standardized ash, sulfur and nitrogen, (2) in using physical properties to distinguish the major classes or orders, (3) in recognizing British thermal unit values in the minor subdivision, (4) in the large number of classes recognized (thirty-six), (5) in proposing mineral names for all classes, (6) in proposing a letter code for the several classes for practical use, (7) in proposing a code for expressing what has commonly been called "grade" of the coal, the grade code to be combined with the rank code to designate and describe a coal. The writer still thinks that the principles then offered, though not necessarily the exact proposals, are valid and desirable in any practical classification.

The actual classification of coal by rank was based on the ratio of fixed carbon to volatile matter for the high-rank coals—the well-known

* State Geologist of Pennsylvania.

and long used "fuel ratio"—and on the ratio of fixed carbon to moisture (as received) for the low-rank coals. This scheme has the advantage of being independent of ash or sulfur content but the range of value is small, being from 1 for lignites to 10 or 12 for hard anthracite.

Since the presentation of that paper in 1919, the whole subject of coal classification has come into the open and been much studied and discussed. The writer has presented several revised versions of his original classification, all seeking to simplify his original proposal, which was generally criticised as too complex.

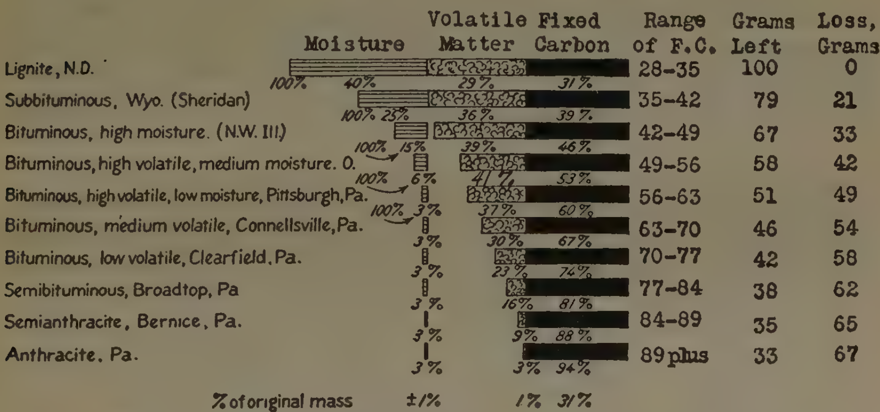


FIG. 1.—LARGER STEPS IN PROCESS OF DEVOLATILIZATION OF COAL.

This chart is drawn on the assumption that in the process of devolatilization there has been no loss in fixed carbon. The percentage of moisture, volatile matter and fixed carbon are characteristic of the coals listed. The several bars are drawn to scale and to show that though the percentage of moisture may remain the same, there is a continuous reduction in the actual amount of both moisture and volatile matter from one class or rank to the next, with a corresponding loss in the total weight of the original mass of coal.

In his original and subsequent papers, the writer has pointed out what appear to him to be significant elements in the problem. Some of these may be repeated here:

1. The "moisture" is an integral part of the coal and must be used in any classification, at least of the low-rank coals.

2. The change from peat to graphite is essentially a process of devolatilization of the coal (including moisture) and may be expressed most fully by using either the increasing percentage of fixed carbon or the decreasing amount of volatile matter (including moisture).

3. Of the various elements composing the coal, whether in the ultimate or proximate analysis, fixed carbon is the only one that maintains a fairly uniform gradient from low-rank to high-rank coals. Volatile matter and British thermal units double back on themselves. Carbon, hydrogen, oxygen and moisture differ but slightly for all of the high-rank coals (see Figs. 1 and 2).

It therefore appears that the simplest and best system for classification will involve the fixed carbon either alone or in ratio with some other element provided that accurate means of determining fixed carbon can be devised. A study of Fig. 2 shows that the ratio of fixed carbon to volatile matter, or the fuel ratio, while serving excellently for high-rank coals, fails completely to distinguish coals below the rank of Pittsburgh coal. On the other hand, fixed carbon divided by moisture, while serving well to distinguish the low-rank coals, is of decreasing value for the high-rank coals. It would seem, therefore, that the best classification will be either to use fixed carbon alone or a combination, as used by the writer 10 years ago, of fixed carbon to volatile matter, above an arbitrary figure, say a fuel ratio of 1.4 per cent. and of fixed carbon to moisture below that figure. This combination gives a range of figures from 1

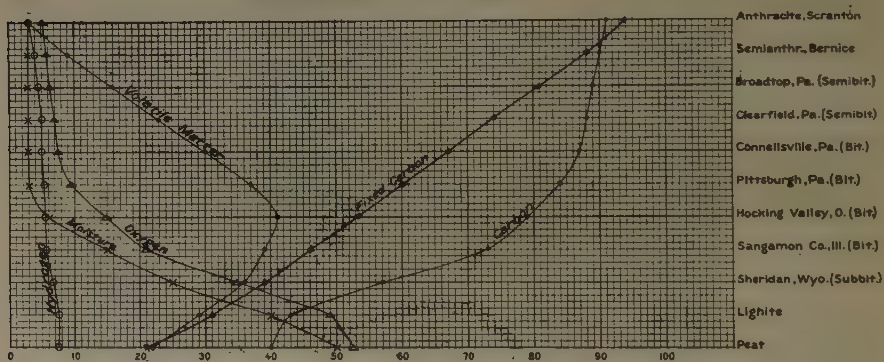


FIG. 2.—VARIATION IN ANALYTICAL ELEMENTS FOR A SELECTED SERIES OF COALS FROM LIGNITE TO ANTHRACITE.

to 12 and requires one computation. Compared with that the use of fixed carbon alone gives a range of from about 20 to 90, and also requires one computation to obtain a figure for the fixed carbon on any standard basis, such as the ash-free basis, or a selected standard-ash basis.

Further, H. J. Rose has recently pointed out that the use of fixed carbon alone lends itself to a simple method of designation by calling coals by number, according to the percentage of fixed carbon on some standard basis. The coal may be further described by using a second figure to express either the "actual" ash or the "actual" fixed carbon. Thus a coal may be called "coal 70," or "coal 70·6," or "coal 65/70," the 6 telling the actual percentage of ash or the 65 telling the actual percentage of fixed carbon in the coal. Having the actual percentage of ash in the coal, the ash-free fixed carbon is found by dividing the actual percentage of fixed carbon by 100 minus the percentage of ash. The actual or guaranteed British thermal units (hundreds omitted) could also be added if desired, thus, written 70·6·140, and read 70 point 6 point 140.

Such additional code letters or figures could be added to express other qualities of the coal as might be agreed upon by the committee on Coal Classification. One advantage of this scheme is that there are no high fences between classes of coal, and the practical man not only knows from its designation exactly what his coal is but is not unnecessarily disturbed if it happens to fall just below the line between larger groups.

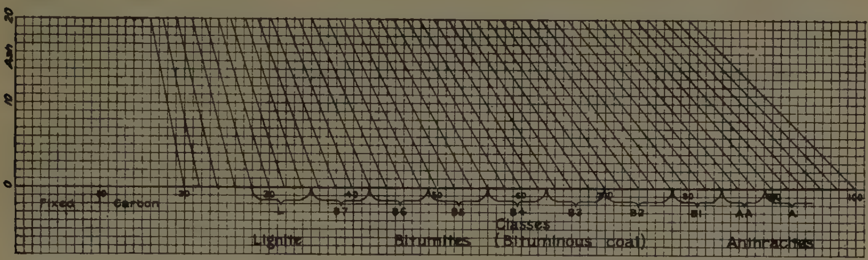


FIG. 3.—CHART FOR DETERMINING THE ASH-FREE FIXED CARBON FOR ANY ACTUAL FIXED CARBON AND ASH.

With this scheme it is not necessary to have so many groups or class names. Indeed, for purely practical purposes, it is only necessary to know the percentage of fixed carbon and ash.

TABLE 1.—*Classification of Coals by Rank*

(Based on Percentage of Fixed Carbon in the As-received Ash-free Coal, and Designated by Number Representing that Percentage; for Example, Coal 56, Coal 70)

Groups and Classes, with Descriptive Notes, and Previous Designations[a]	Coals Included
Anthracites (compact texture; nonluminous flame; fixed carbon, 84 per cent. or higher):	
Class A ("Hard")	89
Class AA ("Soft," "semianthracite")	84 to 88
Bitumites or bituminous coals (compact texture; luminous flame; black streak; fixed carbon, 35 to 83):	
Class B 1 ("Admiralty," "Smokeless," "Loervol")	77 to 83
Class B 2 ("Semibituminous," "Low-volatile," "Lovol")	70 to 76
Class B 3 ("Medium-volatile," "Midvol")	63 to 69
Class B 4 ("High-volatile, low-moisture;" "Hivol")	56 to 62
Class B 5 ("High-volatile, medium-moisture;" "Hiervol")	49 to 55
Class B 6 ("High-volatile, high-moisture;" "Moistvol")	42 to 48
Class B 7 ("Subbituminous," "Black lignite," "Himoist")	35 to 42
Lignites (woody, fibrous, earthy texture; brown streak, buried coals; fixed carbon < 35):	
Class L (well compacted when fresh)	28 to 34
Class LL ("brown coal" of Germany, not well compacted when fresh)	28 to 34
Peat (surface deposits)	

[a] For other than common or humic coals add: K for cannels or canneloids, KK for bogheads, X for splints, and other numbers or letters as needed.

If one is dealing with many coals a simple chart, like Fig. 3, will enable anyone to read off the ash-free fixed carbon. Simply read on the horizontal line the percentage of fixed carbon (actual), then read up for ash, then follow the diagonal line down to the base line.

For scientific purposes it may be desirable to divide coals into a number of classes, somewhat as indicated in Figs. 1 and 2. For that purpose the classification of coal by rank given in Table 1 is suggested for consideration by the Coal Classification Committee, it being understood that the exact limits of classification are suggestions only and subject to future determinations.

For a thoroughly practical classification, it would still be necessary for the Classification Committee to decide upon a system of designating the other qualities of coal aside from fixed carbon, such as size, percentage of ash, British thermal unit value, percentage of sulfur, fusing point of ash, and any other qualities that now or in the future may come to have meaning for some user of coal. At the moment the principal problem, if such a scheme as this is to be adopted, is to secure a better or more standardized method of determining the fixed carbon of coal, and that matter is already under study by a subcommittee on Technical Testing.

[For discussion, see page 554.]

Outline of a Suggested Classification of Coals

By David White, Washington, D. C.

(New York Meeting, February, 1930)

While a country is small and its coal fields are not many, it may be possible to classify its coals on some basis that avoids both overlap and inconsistency, and that may be satisfactory to the particular country, such as geographic distribution or geologic age, or regional uses. In the adjoining country, however, the coals may differ more or less; broad areas may have escaped the erasure of transition zones by erosion, and the stages of alteration of the coals may lead to a classification different from and even incompatible with that of the other country. Such classifications, developed primarily along national lines, are essentially opportunistic. Further, many even of the "use" terms that have been in more common employment with a degree of international application have lost their significance and value, due to changes in industrial methods and urban life.

When the number of coal fields is large and the classification is designed to cover the coals of many countries, the groups or divisions into which the coals are integrated are found to verge one into another; and, though the main divisions may be so naturally composed and oriented that they are relatively distinct and readily separated as entities or units when taken by themselves, the sequence, through classes, groups, and subdivisions, is so complete that we find transition zones through which boundaries must be arbitrarily drawn with minute technical definition. This situation, which is conspicuously illustrated in the United States, is due to the progressive evolution of all coals, irrespective of geologic age, from peats to successively higher ranks, which, if the progressive alteration goes far enough, ends with graphite. This evolution is now firmly established and its complete acceptance is but a matter of time. A classification of all coals in all coal fields can not escape this fact and must take into account all the stages in the transformation of the fuel from rank to rank through division to division, and through group to group.

The plan here outlined aims at a natural classification of coals based on their evolution from peats to graphite. As in other classifications, the differentiation of the various groups and divisions takes account of changes resulting from the progressive losses of moisture and of oxygen of constitution; the relative conservation of carbon; the progressive increase in fixed carbon (inverse to the loss of volatile matter) in

the higher groups, and the at first increasing and later decreasing British thermal unit values, all of which mark the carbonization or so-called "coalification" of the fuels. The plan does not, however, attempt to specify the exact percentages and values which may delineate the boundaries between the different groups, and requires further analytical and calorimetric information regarding some of the divisions. It should not be completed without the cooperation of the industrial users of coal. The terms used are simple. New Greek derivatives will not make any classification more scientific. On the other hand, the more scientific in its method and structure a classification can be made, the more fully it should be adapted to the requirements of a "use" classification.

Outline of Suggested Classification

Viewed in broad perspective from the classification standpoint, coals naturally divide themselves into a number of outstanding and relatively easily recognized major divisions or classes which are distinguished by more or less obvious physical characters, such as color, density, behavior on exposure to air, luster, friability (reflecting the progressive development of cleavage), and recementation, which precedes conchoidal fracture. To these may be added moisture content, another very important character that may be regarded as, in effect, physical. Even some of the subordinate divisions or groups may be characterized by physical features before recourse is had to the analysis or thermal test for supplementary criteria that are more detailed as well as more exact.

In the suggested classification, as shown in Table 1, the entire coal series, beginning with peat and ending in graphite, is divided into five classes, corresponding to the principal natural divisions. These are distinguished mainly by physical characters, though, as with the groups and subdivisions of different orders, the ultimate hard and fast lines of demarcation, which generally fall in transition zones, require arbitrary demarcation on a chemical basis, advantageously combined in some cases, at least, with British thermal unit values on either the ash-free or the pure coal basis.

Peats

The peaty deposits or peats, which comprise the first class, represent the first stage of coal—the first step in the process of coalification. The tissues, spore exines, and other structural debris of plants that escape fungal disintegration and bacterial decomposition during the peat-forming process remain throughout the entire sequence of transformations and are present in graphite of sedimentary origin if the graphite has not been too severely crushed. In fact, the entire coal series is shown to consist of peats in successive stages of alteration

TABLE 1.—*Suggested Plan for Classification of Coals*

Classes	Groups	Subdivisions Common or "Humic"	Types of Coal: Splinty	Types of Coal: Canneloid	Types of Coal: Boghead
I. Peaty (Surface or uncompressed deposits. Distinguished by percentage of water.)	Peats	Light ("immature") Dark ("mature")	Attrital peats	Spore peat	Alga saprocol
II. Brown or lignitic coals (Denser; less water; light to dark brown when freshly mined.)	Unconsolidated lignites	Light Dark	Splint lignites	Cannel lignites	Alga lignites
	Consolidated lignites	Light Dark			
III. Bituminous or "soft" coals (Black, with increasing cleavage, B.t.u., and friability; decreasing moisture.)	Subbituminous (Slacking under shelter)	High moisture Low moisture	Subbituminous splints	Subbituminous cannels	Subbituminous bogheads
	Bituminous (Slacking but slowly, if at all, under shelter)	High volatile Mid volatile Low volatile	Splints	Cannels	Bogheads
	Superbituminous (Dry; maximum friability and B.t.u.)	High volatile Low volatile	Dry splints	Dead cannels	Dead bogheads
IV. Anthracitic or "hard" coals (Recemented; becoming lustrous; decreasing B.t.u.)	Subanthracite (Resembles bituminous coal; decreasing B.t.u.; not friable)	High volatile Low volatile	Types microscopically and physically differentiated, but not distinguished chemically one from another or from the common coals.		
	Anthracite (Conchoidal; lustrous black)	High volatile Mid volatile? Low volatile			
V. Graphitic coals (Metallic gray luster; graphitic streak.)	Subgraphite (Graphitic luster; used mostly for purposes other than fuel)	Hard Friable	Not recognized.		
	Graphite (Sedimentary; graphitic luster and streak. Not a fuel)	Structured Schistose			

or transformation, with gradual eliminations, most important of which are moisture and oxygen.

The light peats, often described as "immature," are usually more distinctly woody than the dark peats, in which the woody tissues of the plants have generally suffered further biochemical decomposition, with consequent further disappearance of woody matter. Accordingly, the organic matter has in most cases reached a slightly more advanced stage of carbonization, with deoxygenation, in the dark or "mature" peat, which, other things being equal, is indicated by a more favorable analysis and calorimetric test.

The transition from surface peats to buried and partly dehydrated brown coals is essentially complete if one examines Pleistocene and late Tertiary deposits in all parts of the world. It grades through the interglacial "peats" of Iowa and the neighboring states, the swamp deposits buried in the Pleistocene coastal terraces, and the but little compressed late Tertiary deposits in Texas and Alaska. Therefore, the location of the boundary between the peats and the brown coals is difficult to draw and requires arbitrary fixation. It may best be located on the basis of the percentage of moisture remaining in the deposit, rather than in terms of thickness of cover or loading, which compresses the peat and induces its initial step in the long process of dehydration.

Brown Coals

The second class embraces the brown coals, including the so-called lignites of the U. S. Geological Survey and the U. S. Bureau of Mines. All are more or less distinctly brown when freshly mined, and their progress in the scale of coalification is marked conspicuously by increasing density and loss of water. Many, at least, of the German brown coal deposits fall into the suggested group of unconsolidated lignites. Those of North Dakota and the Eocene of Texas and Arkansas, which frequently are dense, tough, and relatively little jointed, are covered by the suggested term, Consolidated Lignites. Both the Unconsolidated and the Consolidated Lignites include light and dark phases, which are primarily inherited from a corresponding light or dark peat ancestry, though, as the evolution of the deposits progresses, the light deposits as well as the dark become black as they verge into the lowest group of the bituminous class—that is, the subbituminous coals—as may be seen in the lignites of Eocene age in eastern Montana.

Bituminous Coals

The separation of the class containing the brown coals from the next, the comprehensive class of bituminous or "soft coals," is based primarily on color distinction and follows the present usage of the government

bureaus named. This is most logical as well as convenient, though, as in other classes and groups, the final definite line of exact differentiation is necessarily arbitrary. While high in inherited moisture and oxygen, and consequently low in B.t.u. values, the brown coals are in all these respects superior to their parent peats.

The bituminous class embraces the deposits known for generations in eastern America as "soft coals," a term that antedates the development of the subbituminous coals of the Gulf Coastal plain and the western states. The coals of this class are distinguished from the brown coals by their black color, by the increasing refinement of jointing and cleavage, with consequently increasing friability, and by progressive elimination of moisture and oxygen with volatile matter, with resulting improvement in carbonization and progressive increase in B.t.u. values.

Subbituminous Coals

For the group of lowest rank in this class, which succeeds the lignites, the term "subbituminous," now widely used on this continent, is here retained. The group is marked by slacking when exposed for a time to the air even under shelter, on account of the large percentage of inherited moisture still held by the coal. The group appears to admit of desirable division into a high-moisture or low-rank subdivision, and a low-moisture unit, in which slacking is slower and less harmful. The slacking quality fades very gradually as, concomitant with reduction of moisture, the rank of the coal advances. Slacking occurs even in many of the coals included here in the lower portion of the commonly recognized bituminous group if circumstances are favorable and sufficient time is given. In drawing the arbitrary line separating the subbituminous group from the next higher, the bituminous, account may be taken of the moisture of the ash-free or the oxygen content of the pure coals, with coordinated consideration of the British thermal unit values. The two groups may be separated relatively easily by a line drawn through the thin transition zone in a graph in which total carbons and calorific values of the ash-free coals are the ordinates and abscissa.

The second group in the "soft coal" class is characterized by further dehydration and by rapid refinement of cleavage, with consequent increasing friability, and by growing B.t.u. values caused by oxygen elimination and resulting higher "available hydrogen" in the decreasing volatile matter. Rich in volatile matter, they comprise the typical bituminous coals.

It is generally considered that this group of coals, whose elaboration is receiving special consideration by the American Committees on Coal Classification, calls for subdivision in greater refinement than heretofore in this country, with definitions to accord more nearly with pres-

ent industrial uses. The three subdivisions suggested here may be found adjustable to the industrial viewpoint, especially if the line separating the subbituminous coals be not drawn at too high a rank. On the other hand, if that line is drawn very low a further subdivision may be needed to include coals of an intermediate position.

In passing, it may be remarked that "volatile matter" and "fixed carbon," as determined by the conventional proximate analysis, are of little value in classifying or determining the ranks of peats, lignites, and the lower subbituminous coals, though it is of great value in roughly indicating the stage of coalification (carbonization) of the higher ranks of the bituminous and succeeding groups. In the integration of the coals of the lower groups, total carbon and British thermal units of the ash-free coals are useful, and they are particularly significant in differentiating the lignites from the subbituminous group. The use of fixed carbon in the differentiation of the mid-rank bituminous and the ranks of higher groups ameliorates the need for ultimate analyses in classification, however valuable the latter may be in tracing the evolution of the series.

Superbituminous Class

For the next higher group in the bituminous class, the name "superbituminous" is recommended to supersede the inept term, "semibituminous" now used, while, at the same time, affording opportunity for a desirable slight broadening in scope, with redefinition. The superbituminous coals, typified in the Pocahontas and New River groups, are dry, and, as shown by the ultimate analyses, are marked by continued dehydration and devolatilization, the latter characterized, in particular, by important diminution of oxygen and simultaneous relative enlargement of available hydrogen, which accounts for the increasing British thermal unit values. Calorific efficiency reaches its maximum at or near the top of this group which, consequent to increasing refinement of cleavage, attains maximum friability at the same time. The higher rank coals of this group, which it may be found desirable to separate into high-volatile and low-volatile subdivisions, are strictly superbituminous.

Anthracites

The anthracitic class, the next master division, is synonymous with "hard coals," long in colloquial use in the eastern states. It is physically distinguished from the bituminous or soft coals primarily by recementation of the organic matter of the coal which, above the lower group of the class, becomes lustrous. Recementation appears to mark an important chemical change occurring at the top of the superbituminous group, for above this and throughout the anthracitic group, the British thermal units decrease with decrease in volatile matter.

Subanthracites

The word "subanthracite" is here proposed to designate the coals now generally classed as "semianthracites," the lowest group in the anthracitic class. The group includes the Bernice coals, the Virginia Pocono coals, and similar coals in Arkansas. Physically these coals resemble bituminous coals of the middle ranks, being comparatively coherent, so that in friability they appear to be readily distinguished from the top ranks of the superbituminous group, though differing from the latter by decrease in British thermal units with advancing rank. These features seem rather readily to distinguish the subanthracites from the superbituminous group. The two groups are readily integrated if the coals are graphically plotted with reference to British thermal units and fixed carbon of pure coal.

In view of the considerable range, amounting to approximately 10 per cent, in the fixed carbon of the subanthracites, it is suggested that a division of the group into high-volatile and low-volatile subanthracites may be useful.

True Anthracites

The next higher group in the "hard coal" class embraces the true anthracites, which are characterized by lustrous black color and conchoidal fracture and which become more prominent as recementation becomes more complete.

In view of the segregation of the subanthracites (present semianthracites) as a distinct group, it may not be necessary to subdivide the anthracites further than to differentiate high-volatile and low-volatile ranks. It is suggested, however, that consequent to the advancing depletion of the anthracites of the southern anthracite field, the mining of the coals in the western prongs of that field and of the western middle field may develop the presence of anthracites which, though probably lustrous, may be less conchoidal, higher in volatile, and greater in British thermal units than any of the anthracite coals now mined in Pennsylvania, and which may require an additional and lower subdivision under the anthracite group. It would conform to expectation based on coal evolution if there were somewhere found intergradation from the subanthracite group into the more lustrous anthracites.

Subgraphitic Coals

The coals of Rhode Island, though retaining the fossil structures of the original peat from which they are descended, are now graphitic in luster, with a semigraphitic streak. Further, though they can be used at some trouble as fuels when burned on special grates or mixed with lower rank coals, the principal uses of these graphitic coals are for furnace linings, paints and substitutes for more costly refractories.

Therefore, in spite of the technical propriety of including them as a superanthracite group in the anthracitic class, as has sometimes been done, it would seem more appropriate, in view of their metamorphism and mineral characteristics, to place them, as a low group, in the final class of the coal series—namely, the graphitic coals, which are characterized by metallic gray luster and the graphitic streak. For the group represented by the Rhode Island coals, the term "subgraphitic" is accordingly proposed. Preference as to inclusion of the subgraphitic group in the anthracitic class or the graphitic class may vary with the personal opinions of geologists or engineers.

Graphites

The graphite group, in the final class, contains only sedimentary graphite of such purity or such structure as to warrant regarding it as the descendant of a coal.

Types of Coal

The discussion to this point has to do entirely with coals of the ordinary kinds, which are technically known as "woody" or "humic," on account of the large amount of vascular land plant vegetation and products of the biochemical decomposition of land vegetation which make up the principal plant substance of the deposit. These common coals compose the humic or woody type.

The characteristics of the uncommon or rare types of coal are derived from the special conditions under which the organic sediments are deposited and the kinds of organisms contributed. The microscopical characteristics of these types—spores, algae, etc.—are inherited from the recent deposits. The physical features, though not always apparent in the initial stages, become clearer as the deposit is consolidated and they too continue into the graphitic class. The chemical qualities which distinguish these types also are borrowed from the ingredient matters of the deposit at the beginning. They change, however, with the progressive elimination of volatile matter, especially the oxygen, which was comparatively low at the beginning, and are lost before anthracitization takes place.

Splinty Coals

The splinty type, concerning which considerable confusion prevails, is not always distinguished from the ordinary woody type of coal with which it intergrades. It has recently been defined by Dr. Reinhardt Thiessen,[1] of the U. S. Bureau of Mines, as composed mainly of attrital and spore exine matter, in which relatively little log, branch, and twig wood, humified and jetlike, is to be found. The dense blocklike specimens described by Dr. Thiessen are said to be further characterized by opacity of portions, at least, of the attrital colloid. This opacity, by

[1] See p. 644.

which also he correlates the material with the durain of the British and German coal fields, he attributes to particular original ingredient matters of unknown source.[2]

The lamination, the slightly lustrous and relatively grayish tint, and the frequent layers of fusain ("mother of coal") on the bedding planes of splints immediately distinguish them from the cannels. They show that the splints were deposited under very different conditions which, if the writer is not mistaken, favored the decomposition of the woody matters in an environment in which the surface of the deposit was frequently exposed to the air.

The marketed splints belong to the bituminous group, those best known having a fixed carbon of about 59 to 62 per cent., pure coal basis. The types are recognizable in the subbituminous group and should be readily recognizable in the lignitic group, although they have not received special attention in those groups. In view of the apparent elimination of the chemical characteristics of splints that takes place just before or during the advance of the fuel through the superbituminous ranks, the term "dry splint" is suggested to correspond to the superbituminous group.

Canneloid and Boghead Types

Both the canneloid and the boghead types are aquatic deposits, faintly if at all laminated, and mainly massive and conchoidal in fracture, though sometimes somewhat slabby. The canneloid type, made up predominantly of the exines of spores and pollen grains, verges, by the introduction of occasional algae, into the boghead, which is characterized by fatty algae, mostly of one-celled colonial types, in numbers often so great as to make up the greater part of the deposit. In the first stage the unconsolidated canneloid type of the peat rank is merely a peat made up largely of spore exines. To the corresponding alga deposits, the term Alga Saprocol may be applied, "saprocol" having some years ago been proposed by H. Potonié for this type of recent sediment.

As is well known, the cannels are distinguished even in the early stages of their development by their high volatile, which is richly hydrogenous; and the boghead series, which are extremely fatty, by their extraordinary amounts of very richly hydrogenous volatile matter. The different ranks of these types are not yet, however, distinguished by the characters or amounts of their volatile matter, which varies with the initial composition of the sample. Detailed analytical data showing the sequence of changes are not available. Therefore, though the cannels and

---

[2] The writer is, on the other hand, inclined to regard the opacity as incidental to the rank or stage of transformation of the attrital colloid in the particular areas from which the samples were obtained.

bogheads of the lignitic ranks are generally recognized by their brown color, the ranks of these types through the subbituminous and bituminous groups are at present determinable only by the ranks and groups of the associated coals of the ordinary type.

In the superbituminous group both the cannels and the bogheads appear to have lost not only the characteristically large volume but also the characteristic richness of their volatile matter, which is reduced apparently to the level of the ordinary woody coals, so that while they retain their physical and paleontological characteristics, they lack their distinctive chemical qualities and are, in fact, dead or spent. On this account the terms "dead cannel" and "dead boghead" are here proposed for these types in the superbituminous ranks. Apparently the volatile matter of these types is reduced to the common denominator of the corresponding rank of ordinary coals in the early stages of the devolatilization which marks the superbituminous group.

Though, as already mentioned, the microscopical and physical features of the cannels, bogheads and splints are recognizable through the higher groups, including the subanthracites, the anthracites, and the graphitic coals, if their structure has not been obscured by crushing, these deposits, so far as now known, lack chemical characteristics distinguishing them from the corresponding high ranks of the ordinary coals. Therefore, although a detailed classification coordinate with that of the common coals is of scientific, especially genetic, interest, it appears to be of no economic value aside from the fact that the subanthracites and anthracites of the cannel and boghead types are generally less friable and consequently suffer less waste in handling.

Ultimate Definitions

The demarcation of the precise boundaries between the successive classes, groups, and subdivisions suggested in the foregoing classification may be defined in terms of moisture, oxygen, fixed carbon or total carbon, and British thermal units, using probably only two of these factors. These boundaries should not be drawn without reference to the uses to which the coals are especially fitted. A genuinely scientific classification and a sound and fearlessly drawn use classification should coincide, at least in their main features. Classification bent to meet geographic boundaries or jerrymandered to suit trade names or market abuses can not hope for widespread adoption or permanence.

DISCUSSION

H. G. Turner, Bethlehem, Pa. (written discussion).—Dr. White's suggested classification of coals is a very complete, simple and scientific classification. I am particularly pleased to see the change in the terms semibituminous, semianthracite and superanthracite. These terms were always misleading and ought not to have

been used in the first place. The new terms superbituminous, subanthracite and subgraphite leave no question as to the rank of coal which they designate.

I can see where those who are interested in a use classification may object to the terms "dead cannels" and "dead bogheads." I have heard the term "dead" applied frequently to old weathered coals which have lost a lot of their efficiency through long exposure. Perhaps the terms "former cannel" or "with cannel structure" might be more acceptable

The term "cleavage" has been applied to coal by many writers including myself. I think that "regular fracture" is a better term, for cleavage as used in connection with coal is not the same as cleavage developed in minerals and rocks.

In connection with the subgraphitic coals of Rhode Island, I think the statement that these coals still retain the fossil structures of the original peat is somewhat misleading. The only evidence of plant structures shown in much of this coal is the occasional chip or film of fusain.

This classification is, in my opinion, a firm foundation upon which to build further refinements.

G. H. Cady, Urbana, Ill. (written discussion).—The classification proposed by Dr. White possesses characteristics based upon naturalistic considerations that cannot well be ignored in any scientific classification. It might be called a genetic-metamorphic classification since it recognizes classes in accordance with the stages of metamorphism and types in accordance with the type of organic material from which it was formed. The classification suggests that metamorphism produces changes which eventually obscure or obliterate chemical distinctions between types. To the present writer the suggestion comes that subdivision of coals into two main groups might be possible on the basis of this distinction, one group being that in which division into types is possible, the other one in which division into types is impossible.

The recognition of the so-called splinty type of coal may be based upon valid distinctions. It appears to the present writer, however, that a more definite indication of the nature of the "opaque matter" in such coals is necessary before we can be certain that it actually constitutes a separable and determinable entity. If the distinction is valid, as Dr. Thiessen seems to believe, it is hoped that the nature of the material may be soon definitely established.

Although, for purposes of scientific classification, distinction exists between boghead and canneloid types of coal, practically it is probably a distinction of no great value, and it seems probable that the two types will be combined into one in adapting the scientific classification to practical uses.

The classification does not indicate that there may be coals representing transition between types, that is, xyloid-cannelloid or xyloid-splinty, etc. It seems probable that transition from type to type may exist as well as transition from class to class.

Greater recognition of the constitution of coal is a step in the right direction toward a more naturalistic classification. It is somewhat of a surprise, however, to find associated with such differentiation, separation based upon empirical consideration of the sort providing the basis for separation into groups. Here distinctions of quite a different sort are appealed to than those providing naturalistic separation into types and classes. Particularly does it seem unfortunate to raise distinctions based upon the so-called slacking properties of coal when other distinctions such as heat value, moisture content, and coking properties are equally if not more important and probably better founded on experimental evidence.

The difficulties with the selection of slacking propensity as a basis of naturalistic classification at the present time are several, but chief is the lack of accurate information in regard to the nature of slacking. For instance, "black lignites" have, at least in some instances, the property of breaking or checking when air-dried in the same

manner as gels. Certain of the coals, which apparently Dr. White includes in his subbituminous class, do not possess this characteristic at all. They may break up somewhat by weathering, but this seems to take place along incipient more or less vertical tight seams along which calcite, gypsum, or pyrite "facings" are usually found or along fusain partings. The slacking propensity of the "black lignites" appears to be one thing, that of the high-moisture bituminous coals another. Even distinction based upon moisture content must be exercised with care, as the difference between coals is not great. If slacking is to be used as a basis of distinction, it appears to the writer that it might better be applied to distinguish coals which check upon air-drying from those that do not, than at a higher position in the series. Certainly such coal would be found to slack down much more readily when treated to the accelerated weathering test used by the Bureau of Mines than coals of the true bituminous type.

D. White (written discussion).—The term *spent* cannel—meaning that the cannel had lost its distinctive chemical property—might be used instead of "dead" cannel. So also as to bogheads. Both types retain type characters through more advanced stages of carbonization.

[For additional discussion, see p. 554.]

Status of Coal Classification in Canada*

By R. E. Gilmore,† Ottawa, Ont.

(New York Meeting, February, 1930)

This paper is a revision of a former paper published in mimeograph form by both the Canadian and American coal classification committees, and is now presented for the purpose of acquainting those interested with the present status of coal classification in Canada. The classification employed for "Customs purposes" is detailed, and the classification schemes from which the customs classification was derived are reviewed and discussed. The activities of the Canadian coal classification committee, namely, the Associate Committee of Coal Classification and Analysis of the National Research Council, is also reviewed.

Reserves, Production and Uses of Canadian Coals

The total coal resources of North America are estimated to be 69 per cent. of those of the world, and of this portion, the distribution is 52 per cent. for the United States and 17 per cent. for Canada. Accordingly, Canadian coal resources amount to approximately one-third of those of the United States, and one-sixth of those of the world. As for the geographical distribution, the bulk of the known coal deposits are in the extreme eastern and western portions of the country. Employing the usual, more or less loosely defined, designations for the different classes and subclasses of coal, the distribution of the Canadian coal resources, according to provinces, is as follows:

Maritime Provinces: Nova Scotia and New Brunswick	Bituminous and semibituminous coals.
Ontario (northern part)	Lignite.
Saskatchewan (and Manitoba)	Lignite.
Alberta and British Columbia	Lignites, subbituminous, all kinds of bituminous, semibituminous and semianthracite coals.

Of the total reserves, roughly 92.5 per cent. is credited to the noncoking lower rank subbituminous and lignite groups, the bulk of which are

* Published by direction of the Chief of Division of Fuels and Fuel Testing, by permission of the Director of Mines Branch, Department of Mines, Canada.

† Superintendent, Fuel Research Laboratories, Department of Mines, Canada.

in Alberta. Less than 0.5 per cent. of the total belong to the noncoking higher rank semianthracite and semibituminous group, leaving approximately 7 per cent. in the bituminous coking-coal class. The coal production figures, however, tell a different story. Of the total output of seventeen and one-half million tons (in 1928), thirteen million tons (approximately 74 per cent.) was bituminous; over half of this was mined in Nova Scotia, and the other half was divided about equally between Alberta and British Columbia, with a comparatively small output in New Brunswick. The consumption of this bituminous output is mainly for steam raising and power purposes in railway locomotives and in industrial establishments. In the coal-bearing provinces bituminous coals are also used extensively as a domestic fuel for heating houses and large buildings. The remaining 26 per cent., amounting to four and one-half million tons annually, is mined mostly in Alberta and belongs to the noncoking subbituminous and lignite groups.

On account of the noncoking qualities and the comparative freedom from smoke when burned, lower rank coals find extensive use for household heating, and for this reason are known as domestic coals. They also find extensive use for steam raising and other similar purposes, for which the cost in comparison with that of other coals available is the deciding factor. Although the consumption of Canadian coals for uses other than mentioned is not large, the prospect of a steady increase in the use of bituminous coals is promising, especially for city gas and by-product coke industries; in the glass, ceramic and cement industries; for smithy and other special uses. The greater utilization of Canadian coal for all these uses is the objective of the work of the Fuel Research Laboratories at Ottawa and other fuel-testing and research organizations in Canada. Considerable interest is being taken in furthering a comprehensive use classification for the different groups of (Canadian) coals, more or less in cooperation with the activities of the Use Classification Technical Committee of the American Standards Assn. Technical Committee. The purpose of this paper, however, is to review the status of the coal classification in Canada, mainly in respect to "scientific" classification, rather than to deal further with the general and special characteristics for special uses.

Classification of Canadian Coals

For classifying Canadian coals, the chemical, physical and general characteristics have been used. A great deal of attention has been given to the geological formation and to the general characteristics of the coal deposits in different parts of the country. For classification purposes, more attention has been paid to the chemical than to the physical properties; *i. e.*, appearance, density. The term "general characteristics" is applied to handling, storage and burning qualities, and so forth.

The late Dr. D. B. Dowling, Geologist of the Department of Mines, and Prof. Edgar Stansfield of the Industrial Research Council of Alberta, and formerly Chief Engineering Chemist of the Mines Branch, Fuel Testing Division, were pioneers in the work of classifying Canadian coals. During recent years, J. H. H. Nicolls of the Fuel Research Laboratories and Dr. B. R. MacKay of the Geological Survey, as well as the writer and others, have been giving considerable attention to coal classification matters.

Dowling, in his earlier coal classification work, described the geological formations and certain physical properties, and employed what he termed the "split volatile ratio" involving proximate analyses only. In his later Canadian coal resources publications he adopted the "Twelfth International Geological Congress" classification scheme which, it is to be noted, included his split volatile ratio scheme. For Alberta coals, Stansfield first employed Parr's system of calorific values plotted against volatile matter, using however the calorific values on the as-mined, adjusted-ash basis instead of on the unit-coal basis. In a recent publication[1] by Stansfield and Sutherland, in which some 20 or so of the better known classification schemes are treated, the scheme showing calorific value plotted against the volatile matter, both on the "pure raw coal basis," is stated to give the most satisfactory arrangement for Canadian coals. Nicolls has been studying the most important coal classification schemes based mainly on chemical characteristics, and after fitting in all the reliable analyses on record, he hopes to draw a definite conclusion as to what schemes are most suitable for the coals of Canada, that is, when correlated with the known physical and general characteristics.

Review of Classification Schemes Advanced for Canadian Coals

1. *Dowling's Split Volatile Ratio*[2].—This classification employed the formula

$$\frac{\text{Percentage of fixed carbon} + \frac{1}{2}\text{ percentage of volatile matter}}{\text{Percentage of moisture} + \frac{1}{2}\text{ percentage of volatile matter}}.$$

The percentages were on the air-dried basis, and the division into classes according to this scheme was as follows: Anthracite with split volatile ratios of 15 and above; semianthracite, 13 to 15; anthracitic coal, 10 to 13; high-carbon bituminous, 6 to 10; bituminous 3.5 to 6; low-carbon bituminous, 3 to 3.5; lignitic coal, 2.5 to 3, and lignite, 1.0 to 2.5.

---

[1] E. Stansfield and J. W. Sutherland: The Classification of Canadian Coals. *Canadian Mining and Metallurgical Bull.* (1929) No. 210, 1158.

[2] D. B. Dowling: The Coal Fields of Manitoba, Saskatchewan, Alberta and Eastern British Columbia. Geological Survey, Department of Mines, Canada (1909) *Pub.* 1035, 44.

2. *Twelfth International Geological Congress Classification.*[3]—In this rather comprehensive scheme, the classes and subclasses are differentiated according to volatile matter content, fuel ratio and split volatile ratio, and according to calorific value and mean ultimate analyses. The general burning qualities, and in certain cases the coking and weathering qualities, are also described. This classification, as advanced in 1913, was in part as follows:

Class A:

(1) Burns with short, blue flame; 3 to 5 per cent. volatile matter; fuel ratio 12 and over; calorific value 14,500 to 15,000 B.t.u.; carbon 93 to 95 per cent., hydrogen 2 to 4 per cent., oxygen and nitrogen, 3 to 5 per cent.

(2) Burns with slightly luminous, short flame and little smoke; does not coke; 7 to 12 per cent. volatile matter; fuel ratio 7 to 12; calorific value generally 15,000 to 15,500 B.t.u.; carbon 90 to 93 per cent., hydrogen 4 to 4.5 per cent., oxygen and nitrogen 3 to 5.5 per cent.

Class B:

(1) Burns with short, luminous flame; does not readily coke; volatile matter 12 to 15 per cent.; fuel ratio 4 to 7; calorific value generally 15,200 to 16,000 B.t.u.; carbon 80 to 90 per cent., hydrogen 4.5 to 5 per cent., oxygen and nitrogen 5.5 to 12 per cent.

(2) Burns with luminous flame; generally cokes; volatile matter 12 to 26 per cent.; fuel ratio 1.2 to 7; calorific value 14,000 to 16,000 B.t.u.; carbon 75 to 90 per cent., hydrogen 4.5 to 5.5 per cent., oxygen and nitrogen 6 to 15 per cent.

(3) Burns freely with long flame; withstands weathering but fractures readily and occasionally has moisture content up to 6 per cent.; makes porous tender coke; volatile matter up to 35 per cent.; split volatile ratio 2.5 to 3.3; calorific value 12,000 to 14,000 B.t.u.; carbon 70 to 80 per cent., hydrogen 4.5 to 6 per cent., oxygen and nitrogen 18 to 20 per cent.

Class C:

(1) Burns with long, smoky flame; yields from 30 to 40 per cent. volatile matter on distillation, leaving very porous coke; fracture generally resinous; calorific value 12,000 to 16,000 B.t.u.

Class D:

Generally contains over 6 per cent. moisture; disintegrates on drying; streak brown or yellow; cleavage indistinct.

(1) Moisture in fresh-mined, commercial output up to 20 per cent.; fracture generally conchoidal; drying cracks irregular curved lines; color generally lustrous black, occasionally brown; split

[3] The Coal Resources of the World, 1, x to xii of Preface.

volatile ratio 1.8 to 2.5; calorific value 10,000 to 13,000 B.t.u.; carbon 60 to 75 per cent., hydrogen 6 to 6.5 per cent., oxygen and nitrogen 20 to 30 per cent.

(2) Moisture in commercial output over 20 per cent.; fracture generally earthy and dull; drying cracks generally separate along bedding planes, often with fibrous (woody) structure; color generally brown, sometimes black; calorific value 7,000 to 11,000 B.t.u.; carbon 45 to 65 per cent., hydrogen 6 to 6.8 per cent., oxygen and nitrogen 30 to 45 per cent.

In this classification, according to the writer's understanding, the calorific value ranges and the ultimate analysis ranges are on the dry and ash-free basis, whereas the volatile matter values appear to be on the as-mined basis. The 6 per cent. and over moisture content of Class D is, however, considered to be on the air-dried basis. As will be noticed, the scheme consists of eight divisions of subclasses fitted into four main classes, which according to Dowling[4] conforms to the following nomenclature:

A1, anthracite coal.
A2, semianthracite coal.
B1, anthracitic coal and high-carbon bituminous coal.
B2, bituminous coal.
B3, low-carbon bituminous coal.
C, cannel coal.
D1, lignitic or subbituminous coal.
D2, lignite.

This classification has seemingly not received serious consideration, mainly on account of the lack of proper coordination of the chemical criteria in the individual classes. Despite the fact that a classification like this one with more than two variables is almost sure to run into difficulties, the descriptions of burning qualities and coking properties of the higher rank coals and the recognition of the moisture content and weathering qualities of the lower rank coals is noteworthy, and in this respect the classification has merits. Since its 6 per cent. moisture clause for the lignitic or subbituminous class has been made use of for "Customs purposes," it is repeated here in somewhat abbreviated form, as above.

3. *Stansfield's Classifications.*—Following the scheme of calorific value plotted against volatile matter, as employed by Parr, these classifications deserve special mention on account of their apparently successful application to Canadian coals. His 1925 classification based on the

---

[4] D. B. Dowling: Coal Fields and Coal Resources of Canada. *Mem.* 59, Geological Survey, Canada (1915) viii of Preface.

"moisture as mined" basis with the ash adjusted to a 10 per cent. ash (or 11 per cent. mineral matter) basis was applied to Alberta coals. By this scheme the bituminous coals are divided into "short-flame" and "long-flame" coals, with volatile matter contents below and above 20 per cent. respectively and with the calorific value dividing line at 12,700 (B.t.u. per pound). The subbituminous class ranges from 12,700 to 10,000 B.t.u. with the dividing line between the so-called black and brown lignites at 8200 B.t.u.

The recently published Stansfield and Sutherland scheme also employs the interrelation of the calorific value and the volatile matter content—this time on "pure raw coal" basis, that is, on the coal as mined but calculated to the ash-free basis instead of with adjusted ash as above. Four main classes or ranks; namely, anthracites, bituminous, subbituminous and lignite or brown coal are recognized, the bituminous and subbituminous classes being each divided into three subclasses, *viz.*, "upper, middle and lower." The grouping and nomenclature suggested for Canadian coals is shown in Table 1.

TABLE 1.—*Stansfield and Sutherland* (1929) *Classification*

Rank of Coal	Percentage of Volatile Matter in Pure Raw Coal	Calorific Value on Pure Raw Coal Basis, B.t.u. per Pound
Anthracite	?–8	14,000–16,000
Semianthracite	8–12	14,000–16,000
Upper bituminous	12–20	14,000–16,000
Middle bituminous	20–(30–45)	14,000–16,000
Lower bituminous	30–50	12,750–(14,000–16,000)
Upper subbituminous		11,750–12,750
Middle subbituminous		9,000–11,750
Lower subbituminous		7,000– 9,000
Lignite and brown coal		?–?

CLASSIFICATION FOR CANADIAN "CUSTOMS PURPOSES"

For customs tariff purposes, only three classes of coal are recognized; viz., anthracites, bituminous coals, lignites and lignitic coals.

Anthracite and Bituminous Coals.—Inasmuch as anthracites on one hand and lignites on the other are on the duty-free list, leaving bituminous coals as the only dutiable class, it has been necessary to draw arbitrary dividing lines between these classes. In the above classification, according to the writer's understanding, it is assumed that the anthracite class includes what are generally known as the semianthracites and likewise the bituminous class includes the semibituminous coals. The arbitrary

dividing line between the anthracites and the bituminous coals is, therefore, the dividing line between the semianthracites and the semibituminous subclasses.

The procedure for determining whether a coal is to be classed as anthracite or bituminous is in part as follows: The sample received in the laboratory, which is assumed to be satisfactorily representative of the lot sampled, is subjected to an ordinary proximate analysis according to standard A. S. T. M. procedure. If the residue in the crucible after the determination of volatile matter can be fairly considered as coked or caked or shows distinct signs of agglomeration, the coal shall not be classed as anthracite; but coals that do not coke and which have a fuel ratio (fixed carbon/volatile matter) of 6 or more are for "Customs purposes" to be classed as anthracite.

Lignite and Lignitic Coals.—According to Appraisers' Bulletin 2814 (Department of National Revenue) dated August, 1923, "Lignite and lignitic coals are to be defined as those grades of coal having on the air-dried basis not less than 6 per cent. moisture content."

This definition, which originated through correspondence and conferences between representatives of the Canadian Department of Mines and the Alberta Government with the Customs Department, carried with it the following explanatory notes: (1) "The standard method for air-drying referred to above is that recently adopted by the Provincial Government of Alberta, and by the Department of Mines, Ottawa, and consists of exposing the crushed coal in shallow layers at room temperature in an atmosphere of approximately 60 per cent. humidity; (2) The words lignite or lignitic coals and subbituminous are used as interchangeable, but since the word subbituminous really means under bituminous, or in other words, lower grade than bituminous, all grades lower than true bituminous coals are to be classed as belonging to the large class of lignite coals and may be termed lignites; and (3) The above classification may be further strengthened by applying the most recent classification of Professor Parr of the University of Illinois."

The clause regarding 6 per cent. air-dried basis moisture content, it will be noticed, followed that recommended for Class D coals (lignitic or subbituminous) in the Twelfth International Geological Congress classification, and the reference to Parr's classification mentioned in the third explanatory note referred to the boundary line of 14,000 B.t.u. per pound "unit coal" between bituminous (western type) and subbituminous coals in his 1922 classification. In the writer's opinion, therefore, the definition as employed for customs purposes may be interpreted as comprising those noncoking coals having (*a*) not less than 6 per cent. moisture content when air-dried at room temperature in an atmosphere of approximately 60 per cent. humidity, and (*b*) less than 14,000 B.t.u. per pound on the dry, ash-free (unit coal) basis.

Discussion Regarding Classification of Coals for Customs Purposes

The imports of coal into Canada in 1928 are reported to amount to roughly seventeen and three quarter millions (short) tons. The exports amounted to considerably less than one million tons, so that the surplus of imports over exports was nearly equal to the total production. Of the total imports, roughly three and three quarter million tons, practically all anthracite, was duty free, leaving about fourteen million tons as dutiable bituminous coal.

The distinction between the duty-free anthracites and the dutiable bituminous coals emphasizes the coking properties and independent of the arbitrary fuel-ratio dividing line employed, if the coal in question cokes or shows distinct signs of caking or agglomerating, it is ruled out of the anthracite class. Likewise, with the lower rank coals, since coals in the bituminous class are, by the better known classifications, characterized as coking coals, and the subbituminous as noncoking, the coking property is a factor in deciding whether a coal may be classed as a dutiable bituminous coal or as a subbituminous coal for free entry. Obviously, therefore, with the present arbitrary dividing lines between the noncoking anthracites and the coking bituminous coals on the one hand and the bituminous coals and the noncoking lower rank coals on the other hand, instead of defining first anthracites and then lignitic coals, the feasibility of making one definition only, for the dutiable class, may be considered. Accordingly, bituminous coals may be defined as those coals having slight to distinct coking properties, which when analyzed according to standard laboratory methods show (1) a fuel ratio of fixed carbon divided by volatile matter of less than 6, and (2) calorific values on the dry, ash-free basis, greater than 14,000 B.t.u. per pound.

Although seemingly quite workable, certain complaints arise from time to time regarding the arbitrary dividing lines now employed. One of these is as to whether or not 6 is the proper fuel-ratio dividing line between the semianthracites and the semibituminous coals. According to Ashley's earlier classification (for Pennsylvania coals) and other classifications, the range of 6 to 7, and even 5 to 7, was used, whereas the dividing line recently advanced by Campbell, and adopted by the U. S. Geological Survey, is 5.

The bulk of the anthracite imported from Pennsylvania has a fuel ratio above 10, the recognized dividing line between the so-called true anthracites and the semianthracites. Considerable amounts of the high-rank coals well above the fuel ratio of 6 or 7, and some grades with a fuel ratio as low as 5, all with good physical properties in respect to hardness and friability, come in from the United States. The Scotch anthracites, with physical properties seemingly equally as good as certain of the hard Pennsylvania and Virginia anthracite-like coals, are distinctly

semianthracite as judged by fuel ratio. As for Welsh anthracites, the best quality coals with 3 to 6 per cent. ash, and fair to good handling properties, have fuel ratios of 10 to 11 and higher, whereas a considerable part of the Welsh coal imported as anthracite falls appreciably below the fuel-ratio line of 10, and in certain cases is dangerously near the dividing line of 6. For these coals, as the fuel ratio drops below 10, the friability becomes poorer, and signs of agglomerating and coking appear for coals with a fuel ratio in the neighborhood of 6. It is for this reason that there is a tendency to class the more friable Welsh coals into the low-volatile bituminous steam-coal group, instead of into a subclass of the main anthracite class; in fact, in the trade there has been an expressed desire to consider the Welsh so-called semianthracites as a class entirely separate from anthracite entitled to free entry as a domestic (household) fuel.

The arbitrary dividing line of 6 per cent. air-dried moisutre content between the lignite and subbituminous coals on the one hand and the bituminous coals proper on the other as now employed for customs purposes has, the writer understands, been serving its purpose fairly well. In the province of Alberta, there are certain areas containing coals with an air-dried moisture content both slightly below and above this line, though seemingly coals from individual mines in these areas are apparently definitely either above or below the line in question. The writer's attention, however, has been drawn to a situation in the state of Washington, where from a certain mine are produced coals with air-dried moisture contents on both sides of the 6 per cent. dividing line, for which coals it has also been noted that the 14,000 B.t.u. per pound (dry, ash-free basis) figure coincides fairly closely with the dividing line of 6 per cent. air-dried moisture content. In other words, from a particular mine may be found coals that require differentiation when applying the customs classification as now in force.

Although not strictly necessary, it is strongly desirable that a customs classification should conform to a recognized scientific, commercial classification, and it is hoped that the efforts of the coal classification committees now active will succeed in arriving at a classification that can be adopted internationally, at least by Canada and the United States, and be of service for customs purposes as well as for scientific and industrial use.

Nomenclature

Application of the recently advanced and more reliable classification schemes to Canadian coals reveals the fact that what have been known in Canada as lignite and lignitic coals by certain of the older classifications come in the so-called subbituminous class, and certain of the subbituminous coals may be grouped in the low-carbon bituminous class. Just

how such a shift would affect the present customs classification remains to be seen, but it is quite possible that a change in nomenclature rather than a change in a dividing line would meet the situation. For example, if it is desired to maintain the present status in respect to the arbitrary dividing line between the lower rank dutiable and duty-free classes of coal instead of defining "lignite coals" as now attempted, all that would be necessary would be to drop the names "lignite and lignitic coals" and substitute the wording "all coals having less than 6 per cent. moisture on the air-dried basis," etc. Hence efforts to obtain a classification scheme suitable for both Canadian and American coals may proceed irrespective of that now employed for customs purposes, as any changes that may be desired may be fitted later into an internationally acceptable coal classification.

As for the nomenclature to be finally adopted for the different classes and groups of coals, the writer is quite in sympathy with the desire of the executive of the A. E. S. Sectional Committee on the Classification of Coal to delay serious discussion until the number of classes and groups are decided upon, but at this stage would like to insert some comments on these, and in particular to the prefix "semi" as used with the terms semianthracite and semibituminous. Although seemingly as difficult to effect a change in these terms as it is to change one's name, yet notwithstanding, it is generally agreed among those in Canada most interested in coal classification matters that a change should be made in either or both of these names. This is in accord with the viewpoint expressed by Stansfield and Sutherland, who, in their recently published classification, suggested for Canadian coals as detailed above, use for both the bituminous and subbituminous classes the terms "upper," "middle" and "lower" to differentiate three groups in each class.

Canadian Coal Classification Committee

The Associate Committee on Coal Classification and Analysis of the National Research Council was formed early in 1928, since which time it has been active in coal classification matters. This committee traces its origin to the attendance by special invitation of several representatives from Canada to the A. E. S. C. preliminary meeting held at the time of the First International Conference on Bituminous Coal in Pittsburgh, November, 1926, out of which meeting grew the present Sectional Committee on Classification of Coal (under sponsorship of American Society for Testing Materials, and rules of American Engineering Standards Committee). During the interval between this time and the first general meeting of the Canadian committee, the writer was the single Canadian representative, as a member at large to the American sectional committee. Realizing that Canada's interests were sufficiently different from those

of the United States to render a separate committee highly desirable, a Canadian associate committee was appointed by the National Research Council, the cooperating organizations being the Alberta Research Council and the Dominion Department of Mines. The function of this committee is twofold; namely, to make a special study and report on coal classification matters peculiar to Canada, and to work in close cooperation with the American sectional committee. By the interchange of representatives and reports of the Canadian and American committees, each committee is kept informed of the work of the other, thus to further the mutual purpose of working out a system of classification applicable to the coals of the North American continent. The meetings of the Canadian coal classification committee are held in different parts of the country, and are generally timed to coincide with annual or semi-annual meetings of technical societies or conventions. The first general organization meeting was held in London, Ontario in June, 1928, at the time of the Annual Convention of Canadian Chemists, and the second general meeting in October, 1929, at Edmonton, Alberta, in conjunction with the western Annual Meeting of the Canadian Mining Institute.

The personnel of the Associate Committee on Coal Classification and Analysis is as follows:

Joint Chairmen:

- Dr. H. M. Tory, President, National Research Council, Ottawa, Canada.
- Dr. Chas. Camsell, Deputy Minister, Department of Mines, Ottawa, Ont.

Secretary:

- F. E. Lathe, Technical Assistant to President, National Research Council, Ottawa, Ont.

Members:

- F. A. Combe, Consulting Combustion and Steam Engineer, 1188 Phillips Square, Montreal, Que.
- J. B. DeHart, Inspector of Mines, Alberta Mines Branch, Lethbridge, Alta.
- J. R. Donald, Director, J. T. Donald Co., 40–42 Belmont St., Montreal, Que.
- George Drummond, Consulting Engineer, Drummond Company, McGill Bldg., McGill St., Montreal, Que.
- G. S. Eldridge, Consulting Engineer, Cave Building, 567 Hornby St., Vancouver, B. C.
- A. E. Flynn, Department of Mining Engineering, Nova Scotia Technical College, Halifax, N. S.
- John D. Galloway, Provincial Mineralogist, Department of Mines, Victoria, B. C.
- R. E. Gilmore, Superintendent, Fuel Research Laboratories, Department of Mines, Booth St., Ottawa, Ont.
- F. W. Gray, Asst. General Manager, British Empire Steel Corpn., Sydney, N. S.
- Charles R. Hazen, Vice-President, Milton Hersey Co., 980 St. Antoine St., Montreal, Que.
- B. R. MacKay, Geologist, Geological Survey, Department of Mines, Ottawa, Ont.
- A. Mailhiot, Professor of Geology, l'Ecole Polytechnique, Montreal, Que.
- John McLeish, Director, Mines Branch, Department of Mines, Ottawa, Ont.
- W. E. McMullen, Deputy Minister, Department of Lands and Mines, Fredericton, N. B.

T. M. Molloy, Commissioner of Labour and Industries, Regina, Sask.
J. H. H. Nicolls, Chemist, Fuel Research Laboratories, Department of Mines, Ottawa, Ont.
L. J. Rogers, Assistant Professor of Chemistry, University of Toronto, Toronto, Ont.
F. H. Sexton, President, Nova Scotia Technical College, Halifax, N. S.
Edgar Stansfield, Professor, Industrial Research Department, University of Alberta, Edmonton, Alta.
L. R. Thomson, Consulting Engineer, New Birks Building, Cathcart St., Montreal, Que.

Three subcommittees were organized with the membership enlarged to include men other than on the main committee, as follows:

Origin, Constitution, Location and Occurrence of Coal.
Chairman:
B. R. MacKay.
Members:
W. A. Bell, W. S. Dyer, J. D. Galloway, I. W. Jones, T. M. Molloy and E. S. Moore.

Applicability of Proposed Scientific Classification Schemes to Canadian Coals.
Chairman:
Edgar Stansfield.
Members:
J. A. Allan, G. S. Eldridge, R. E. Gilmore, B. R. MacKay, J. H. H. Nicolls, F. H. Sexton and J. W. Sutherland.

Sampling, Analysis and General Uses of Canadian Coals.
Chairman:
R. E. Gilmore.
Members (Analysis and Sampling Section):
J. R. Donald, G. S. Eldridge, C. R. Hazen, J. H. H. Nicolls, L. J. Rogers and E. Stansfield.
(Uses of Canadian Coals Section):
C. E. Baltzer, J. B. DeHart, A. E. Flynn, J. D. Galloway, F. W. Gray, G. H. Jenkins, F. G. Neate, R. A. Strong and L. R. Thomson.

For the twofold purpose of furthering the use of Canadian coals for different specified purposes, and of being of service to the Use Classification Technical Committee of the American Sectional Committee, individual uses have been assigned to the special attention of various members as follows:

Domestic bituminous coal, east and west: F. W. Gray, Sydney, N. S., and J. D. Galloway, Victoria, B. C. respectively.

"Domestic" subbituminous and lignite coals: J. B. DeHart, Lethbridge, Alta.

Domestic anthracite from United States and British Isles: L. R. Thomson, Montreal, Que.

Coal for different types of railway locomotives: G. H. Jenkins, Montreal, Que.

Coal for steam generation in pulverized form and coal for domestic coke and city gas: C. E. Baltzer and R. A. Strong respectively, both of the Fuel Research Laboratories, Department of Mines, Ottawa, Ont., Canada.

[For discussion, see page 554.]

Multibasic Coal Charts

By Harold J. Rose,* Pittsburgh, Pa.

(New York Meeting, February, 1930)

One picture is better than ten thousand words.— Old Chinese Proverb.

Graphic methods have long been used to advantage in dealing with diverse phases of fuel technology and research. Not only are graphs convenient for presenting data and making calculations, but they are invaluable for deriving generalizations from a mass of data. This is particularly true when relationships are obscure and numerical correlation is poor.

This paper describes a new form of graph, the Multibasic Coal Chart, which has been developed to facilitate the study and comparison of coals of all ranks from peat to anthracite on any moisture and purity basis. This type of chart is of unique value for coal classification purposes, since it permits a direct visual comparison of the results obtained when coals are classified by the various systems which have been proposed.

Coals are highly variable in character as a result of wide differences in original vegetal ingredients, degree of metamorphism, and nature and amount of associated impurities. For nearly 100 years, attempts have been made to devise a satisfactory method of classification, and many tables, formulas, ratios and graphs have been suggested. Many of the suggested systems possess considerable merit and have found a limited use, thus promoting a better understanding of coal. However, objections have been raised against each system, and none has received general adoption. When the various systems of classification are compared, it is found that there is an apparently almost hopeless confusion due to:

1. Serious lack of agreement as to the proper moisture and purity basis on which to classify coal. More than ten different bases have been proposed, each of which requires complete recalculation of the analysis data used.

2. Lack of agreement as to what chemical and physical properties are most important and best to use for classification purposes. There is too little knowledge of the degree to which these properties can be correlated.

* Director, General Laboratory Dept., The Koppers Research Corpn.; Chairman A. S. T. M. Technical Committee on Scientific Classification of Coal.

3. There is a needless variety in the graphic methods used and frequent violation of the basic principles of graphic presentation.

By using multibasic coal charts, it is possible to bring order out of this chaos to a remarkable extent.

Construction of Multibasic Coal Charts

The simplest and most general form of the multibasic coal chart is shown by Fig. 1. It is seen to consist of two distinct but adjoining areas.

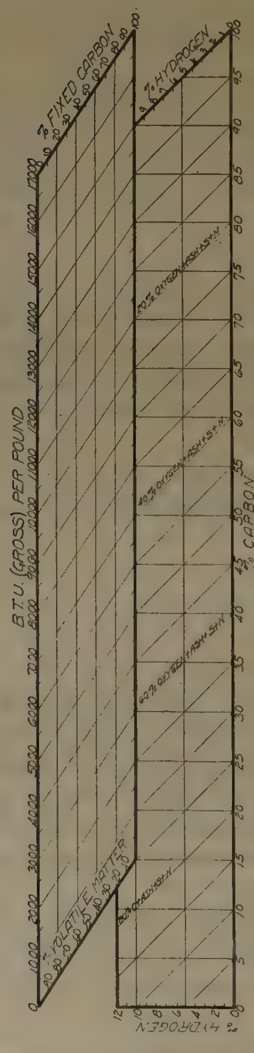


Fig. 1.—Multibasic coal chart No. 12. General diagram showing construction.

Fig. 2.—Multibasic coal chart No. 1. General diagram showing construction of condensed form.

In the lower area, coals may be graphed according to their ultimate analyses, the scales ranging from 0 to 100 per cent. carbon and 0 to 10

per cent. hydrogen (0 to 12 per cent. at left). In the upper area, calorific values and volatile matter or fixed carbon may be presented, the scales ranging from 0 to 17,000 B.t.u. and 0 to 100 per cent. volatile matter (or 100 to 0 per cent. fixed carbon).

If it is desired to use fixed carbon and not volatile matter, the two areas may be fitted together as in Fig. 2 to conserve space and to bring the data in the two areas as close together as possible for convenient comparison.

The lower area may be considered part of a triaxial diagram in which the 45° diagonals represent all constituents other than carbon and hydrogen, which are necessary to make the analysis add up to 100 per cent. Such constituents may be oxygen, ash, sulfur or nitrogen, or any combination of these, depending upon the purity basis employed. (See Figs. 1 and 2.) It is not considered necessary to go further into a detailed mathematical explanation of the chart.

Graphic Study of Various Moisture and Purity Bases

The remarkable properties of the multibasic coal chart become evident as soon as analyses are graphed. On Fig. 3, an analysis of Illinois coal has been presented on nine different purity bases, according to both B.t.u. vs. fixed carbon and carbon vs. hydrogen. This coal was obtained from the No. 5 seam, Peoria County, Hanna City, Ill., and had the following analysis "as received":

Table 1.—*As Received Analysis of U. S. Bureau of Mines Sample No. 22,986*

[Data from U. S. Bureau of Mines *Bulletin* 123, page 35]

Proximate Analysis	Per Cent.	*Ultimate Analysis*	Per Cent.
Moisture	15.41	Carbon	57.96
Volatile matter	34.34	Hydrogen	5.65
Fixed carbon	38.52	Oxygen	20.57
Ash	11.73	Nitrogen	1.12
		Sulfur	2.97
B.t.u. per lb	10,422		
Air-drying loss, per cent	8.9		

On Fig. 3, the above "as received" analysis is shown by the two circles numbered 1, which are seen to be similarly placed in their respective areas. If the analyses are calculated to the "air-dry" basis, both points move about the same direction and distance to 2. If calculated to the "dry (105° C.)" basis, they move still further to points 3. However, if the original analysis had been calculated to the "ash-free" basis, without eliminating moisture, the resulting figures would be represented by points 4. Similarly the pairs of points numbered 5, 6, 7, 8 and 9,

represent, respectively, the same analysis recalculated to the "air-dry ash-free," "dry ash-free," "dry, ash and sulfur-free (Parr corrections)," "dry, ash and sulfur-free," and "dry, ash, sulfur and nitrogen-free" bases. It will be noted that in each case, the point representing carbon

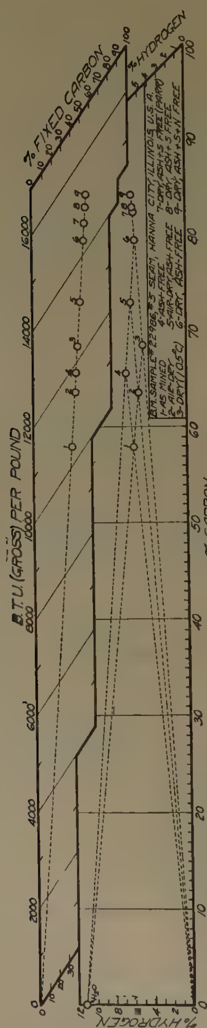


FIG. 3.—MULTIBASIC COAL CHART No. 2. COAL ANALYSIS PRESENTED ON VARIOUS BASES.

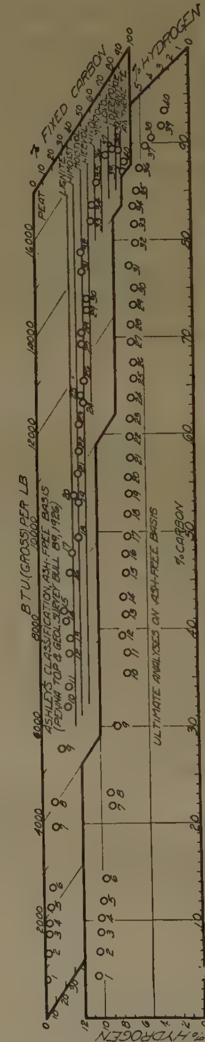


FIG. 4.—MULTIBASIC COAL CHART No. 3. ASHLEY CLASSIFICATION VS. ASH-FREE ULTIMATE ANALYSES.

and hydrogen falls nearly under the point representing B.t.u. and fixed carbon for the same purity basis.

It will also be seen that in the upper area of Fig. 3, all points appear to fall on a straight line originating from the zero B.t.u., zero fixed-carbon corner. As a matter of fact, all points do lie exactly on this line with the exception of 7, which represents "unit coal" and was calculated by using

the special Parr corrections,[1] and deviates slightly, though not enough to be distinctly shown by the graph. The Parr corrections give a somewhat lower B.t.u. value than point 8, which represents the ordinary "dry, ash and sulfur-free" basis. It is hardly necessary to point out the advantages which follow from recognizing the above straight-line relationship, which to the writer's knowledge has never been used before in coal classification work.[2]

In the lower area of Fig. 3, a similar though less simple relationship exists, which is made clear by the dotted lines. Point 1 represents the carbon and hydrogen content of the coal sample "as received." When the analysis is calculated to the "air-dry" and "dry (105° C.)" bases, it is represented by points 2 and 3 respectively, which lie on a straight line originating from the point on the left axis representing 100 per cent. H_2O (11.19 per cent. hydrogen, 0 per cent. carbon and 88.81 per cent. oxygen).

If, however, an analysis is calculated free of ash, sulfur or nitrogen, the point moves on a straight line directly away from the lower left-hand corner, which represents 100 per cent. of these constituents. Point 7, obtained by using the Parr corrections for ash and sulfur, is slightly off the line representing dry coal, but otherwise close to point 8, representing the ordinary "dry, ash and sulfur-free" basis. The deviation between these two values will depend upon the amounts of ash and sulfur present in a given coal sample.

Graphic Study and Classification of Coals of Different Ranks

The foregoing discussion has referred only to the analysis of a single coal when calculated to different moisture and purity bases. What relationships will be found when the analyses of many different kinds of coal are presented on the multibasic coal chart? The answer can be found from Figs. 4, 5, and 6.

For the purpose of this example, 40 analyses of peat, German brown coal, lignite, subbituminous, bituminous, semibituminous coal, semianthracite and anthracite were assembled, the descriptions being given in Table 2. These analyses were selected practically at random, the method being to use the first analyses found which had total carbon (ash-free basis) approximating even numbers, which did not have excessive ash or sulfur contents, and which were not weathered samples.

This method of selecting analyses explains the relatively even spacing of points in the lower area of Fig. 4. There are a few gaps below lignitic rank, for which analyses were not at hand.

---

[1] S. W. Parr: The Classification of Coal. Univ. Ill. Eng. Expt. Sta. *Bull.* No. 180 (1928).

[2] Described in the writer's unpublished reports of March 28, 1929, and Nov. 22, 1929, to the Technical Committee on Scientific Classification of Coal.

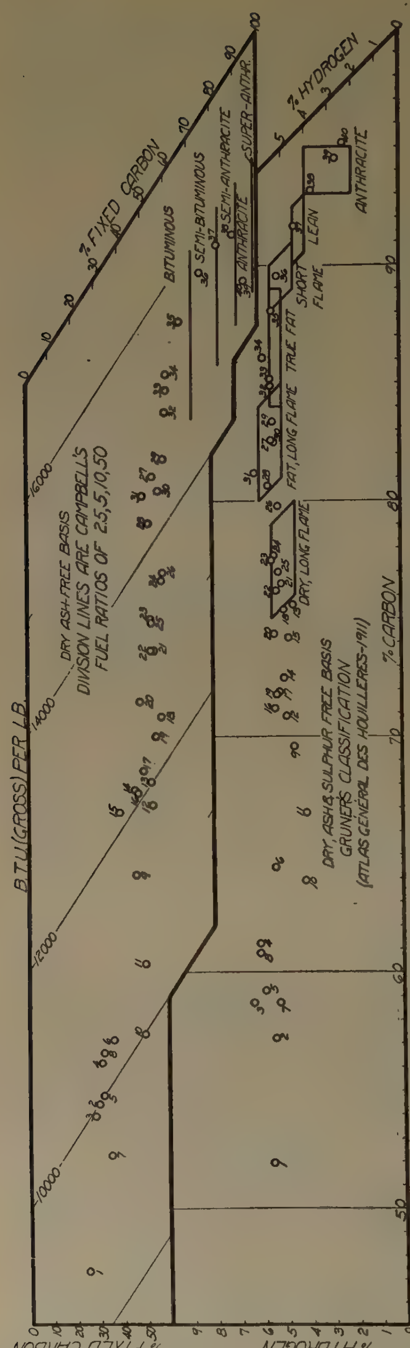


FIG. 5.—MULTIBASIC COAL CHART No. 4A. CAMPBELL FUEL RATIO VS. GRUNER CLASSIFICATION.

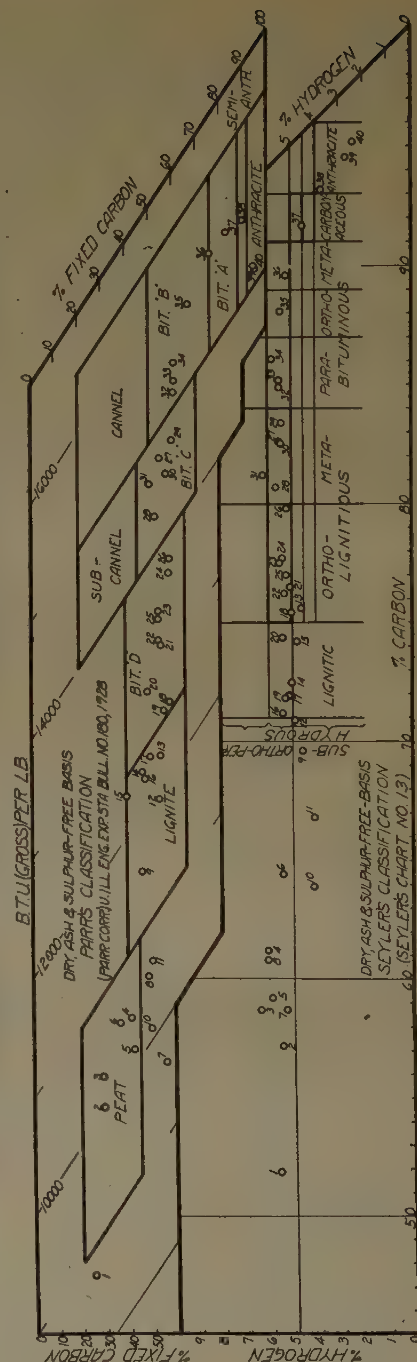


FIG. 6.—MULTIBASIC COAL CHART No. 5A. PARR VS. SEYLER CLASSIFICATION.

TABLE 2.—*Description of 40 Samples of Coal (Figs. 4, 5, 6)*

Key	Sample No.	Description	State	County	Mine	Seam
1	5994[a]	Peat	Me.	Somerset	Pittsfield	
2	5983[a]	Peat	Me.	Kennebec	West Sidney	
3	A39765[f]	Peat	Wis.	Manitowoc	Manitowoc	Hawk Island Bog
4	A39762[f]	Peat	Wis.	Manitowoc	Manitowoc	Hawk Island Bog
5	A39759[f]	Peat	Wis.	Manitowoc	Manitowoc	Hawk Island Bog
6	6027[a]	Peat	Fla.	Duval	Bayard	
7	A39744[f]	Peat	Wis.	Manitowoc	Manitowoc	Hawk Island Bog
8	A39763[f]	Peat	Wis.	Manitowoc	Manitowoc	Hawk Island Bog
9	16355[g]	Brown coal	Germany	Rheinland		
10	11006[c]	Lignite	Mont.	Valley	Jones	Jones
11	10900[c]	Lignite	Mont.	Valley	Young	Young
12	29251[e]	Lignite	N. D.	Williams	Pittsley	
13	12453[c]	Lignite	S. D.	Perkins	Jones	
14	14729[c]	Lignite	N. D.	Morton	Jones	Haynes (?)
15	3812[b]	Lignite	Mont.	Dawson	Snyder	
16	29564[e]	Subbituminous	Wash.	Lewis	Fords Prairie	
17	14754[c]	Subbituminous	Wyo.	Converse	Inez	
18	10740[c]	Subbituminous	Wyo.	Converse	Onyon	
19	9096[b]	Subbituminous	Wash.	Thurston	Hanaford No. 1	
20	12601[c]	Subbituminous	Colo.	Jackson	Riach	Riach
21	17247[c]	Subbituminous	Wyo.	Freemont	Hickey	
22	17248[c]	Subbituminous	Wyo.	Freemont	Mitchell	
23	11740[c]	Subbituminous	Wyo.	Uinta	Kemmerer	
24	15237[c]	Subbituminous	Wyo.	Hot Springs	Big Horn	Gebo
25	14909[c]	Bituminous	Colo.	Moffat	Shafer	
26	29004[e]	Subbituminous	Mont.	Musselshell	Roundup A	Roundup
27	31299[e]	Bituminous	Ind.	Greene	Gilmour No. 7	No. 4
28	19163[d]	Subbituminous	N. M.	M'Kinley	Carreto	Otero (?)
29	30486[e]	Bituminous	Ill.	Franklin	Sesser No. 1	No. 6
30	31050[e]	Bituminous	Ill.	Franklin	Orient	No. 6
31	19844[d]	Bituminous	Utah	Carbon	Castlegate No. 1	D
32	15675[c]	Bituminous	Ohio	Jefferson	La Belle	Lower Freeport
33	24714[d]	Bituminous	Ky.	Harlan	Coxton	Harlan
34	21644[d]	Bituminous	Tenn.	Anderson	Klondike or No. 5	Coal Creek
35	19641[d]	Bituminous	Ala.	Jefferson	Pratt No. 4	Pratt
36	12370[c]	Bituminous	Pa.	Cambria	Logan No. 5	Miller
37	20599[d]	Semibituminous	W. Va.	McDowell	No. 1	Pocahontas No. 4
38	30697[e]	Semianthracite	Va.	Montgomery	Merrimac	"Big Vein"
39	11441[c]	Anthracite	Pa.	Luzerne	Colliery No. 14	Pittston
40	11782[c]	Anthracite	Pa.	Luzerne	Dorrance	Red Ash

Source of analyses: [a] Bur. Mines *Bull.* No. 16; [b] Bur. Mines *Bull.* No. 22; [c] Bur. Mines *Bull.* No. 85; [d] Bur. Mines *Bull.* No. 123; [e] Bur. Mines *Bull.* No. 193; [f] Bur. Mines analyses obtained through courtesy of Dr. R. Thiessen; [g] unpublished analysis, Koppers Research Corpn.

The value of the multibasic coal chart is forcibly brought out when the location of the 40 analyses in the upper and lower areas of Fig. 4 is compared. Throughout the whole range of rank from peat to semibituminous coal, the points representing carbon and hydrogen for each coal (lower area) are approximately below the points in the upper area representing B.t.u. and fixed-carbon of the same coals. A direct correlation in the placement of points does not hold for semianthracite and anthracite, owing to the fact that such coals decrease in B.t.u. (due to loss of hydrogen) as their carbon content increases. That is, in the upper area of Fig. 4, points 38, 39 and 40 trend downwards and to the left towards pure carbon,

which would be located at about 14,500 B.t.u. and 100 per cent. fixed carbon, while in the lower area points 38, 39 and 40 trend downwards and to the right towards the 100 per cent. carbon corner. The reverse trend or "hook" in the B.t.u. vs. fixed carbon curve for anthracites is characteristic, and does not materially reduce the value of the chart.

The manner in which the multibasic coal chart may be used to compare different classification systems is illustrated by Figs. 4, 5 and 6, which show the same 40 coals classified according to proposals of Ashley, Campbell, Gruner, Parr and Seyler. The various purity bases used are indicated on each chart.

The upper area of Fig. 4 illustrates Ashley's classification, based on ash-free coal containing as-mined moisture. The coals are divided only with respect to fixed carbon (7 per cent. intervals being used in most cases), and the classes are seen to consist graphically of long parallel strips without any use of B.t.u. values for cross classification.

Since none of the analyses presented in Figs. 5 and 6 contain moisture, the left ends of the multibasic coal charts (below 45 per cent. carbon) have been omitted. The upper area of Fig. 5 shows the same 40 coals recalculated to the dry, ash-free basis. The extent to which the dry basis fails to place ash-free coals in the same order as the as-received moisture basis, can be seen by comparing with Fig. 4. The four division lines and names at the right refer to the fuel ratios of 2.5, 5, 10 and 50 which Campbell[3] has proposed as part of a basis of classification, his criteria for coal of lower than bituminous rank not being shown.

The lower area of Fig. 5 refers to the Gruner classification, which has been extensively used in continental Europe. The analyses have been calculated moisture, ash and sulfur-free. The six areas represent the six classes and names proposed by Gruner. The degree to which these classes overlap, or fail to include some coal analyses, is illustrated. Since the analyses are sulfur-free, they are all moved slightly to the right, with respect to the dry, ash-free analyses in the upper area of Fig. 5. The degree of this displacement varies with each coal, depending upon the original sulfur content.

Fig. 6 compares the Parr classification with that of Seyler,[4] which has recently been considerably used by British fuel investigators. Both of these systems, like that of Gruner, are based on dry, ash and sulfur-free coal, but Parr uses a special method of calculation designed to compensate for the loss of water of hydration of mineral impurities and change in weight of pyrites during the ash determination.

---

[3] M. R. Campbell: Our Coal Supply—Its Quantity, Quality and Distribution. *Proc.* 1st. Internatl. Conf. Bituminous Coal, Pittsburgh, Pa. (1926).

[4] C. A. Seyler: The Commercial Classification of Coal. *Proc.* So. Wales Inst. Eng. (1900) **21,** 483; also The Chemical Classification of Coal. *Fuel in Science & Practice* (1924) **3,** 15–26, 41–49, 79–83.

With the exception of coals 1, 10 and 11, the 40 coals fit neatly into the divisions which Parr has selected. The Seyler classification provides more subdivisions than most other systems. The boxes outlined with the thicker lines are considered by Seyler to represent the common or typical coals, while areas of higher and lower hydrogen content provide space for less common kinds of coal.

Ratios such as hydrogen:oxygen, carbon:hydrogen, and carbon:oxygen have also been proposed for coal classification, and can be shown in a simple way in the lower area of the multibasic coal charts. A graphic study of these ratios has been made by Rose,[5] who pointed out certain of their limitations. In a similar way, the locus of the Goutal formula for calculating B.t.u. from proximate analysis can be shown in the upper area.

The effect of oxidation and/or heat upon the analysis of coal has been shown by Rose and Sebastian[6,7] on the multibasic coal chart. Many other relationships can be shown. Differences in original vegetal composition may cause a vertical displacement[8] of points from the narrow band representing typical coals of predominantly humic character. Thus algal and cannel coals are characteristically rich in hydrogen and low in fixed carbon.

Since on the "dry, ash-free basis," volatile matter + fixed carbon = 100 per cent., a single point will represent both of these variables in the upper area. However, on the "ash-free basis," volatile matter + moisture + fixed carbon = 100 per cent. In this case volatile matter and fixed carbon must each be represented by a separate point, and the vertical distance between these points represent the moisture percentage. This provides a convenient means of representing the whole ash-free proximate analysis in the upper area. Studies of this sort have been made, and may be presented in a later paper.

Presentation of Physical Data

We have now seen how it is possible, on a multibasic coal chart, to graph accurately carbon, hydrogen, oxygen, B.t.u., volatile matter, fixed carbon and moisture, and to show simultaneously the locus of various ratios and formulas. We have seen how it is possible to present analyt-

---

[5] H. J. Rose: Selection of Coals for the Manufacture of Coke. *Trans.* A. I. M. E. (1926) **74**, 600. Also available as private reprint, with statistics revised to 1928.

[6] H. J. Rose and J. J. S. Sebastian: Changes in Properties of Coking Coals Due to Moderate Oxidation During Storage. See p. 569.

[7] H. J. Rose and J. J. S. Sebastian: Coal Storage: Effect of Oxidation and Heat on the Analysis of Bituminous Coal. Presented before Gas and Fuel Div., Amer. Chem. Soc., Minneapolis Meeting, Sept., 1929.

[8] D. White: Progressive Regional Carbonization of Coals. *Trans.* A. I. M. E (1925) **71**, 276.

ical data on any desired purity basis, and thus compare various systems of coal classification. Furthermore, it is evident that the trend of points from left to right shows the orderly progression in rank due to metamorphism, while differences in original vegetal composition or the effects of oxidation or heat are reflected by vertical or oblique displacement of points from the narrow band of typical coal analyses.

However useful the above combination of advantages, the multibasic coal chart would still be of too limited scope unless it could be used to express physical characteristics of the coals as well. This can readily be done in several different ways.

For presenting qualitative data the common expedient of using different symbols or colors may be followed. For example, coking coals may be represented by circles, badly slacking coals by crosses, etc. When numerical values are available, those numbers may be written down beside the points representing the chemical analysis.

In most cases however, it will be found desirable actually to graph the results of physical tests. This is readily possible. In the case of multibasic coal chart No. 10, which is ready for distribution but is not illustrated in this paper, a third area with ordinary cross-section ruling is located below, but adjoining the chart. In this area it is possible to graph such numerical data as shatter test, accelerated slacking test, agglutinating value, softening temperature, specific gravity, moisture, etc. Such values are placed below the corresponding ultimate analysis points, and permit instant visual comparison with the data in the two areas lying above.

However, it is not always necessary to use a third cross-section area to graph physical data. For example, multibasic coal chart No. 11 which is ready for distribution, and which is a much enlarged version of Fig. 1, contains 100 vertical (volatile matter or fixed carbon) divisions in the upper area, and 100 vertical (hydrogen) divisions in the lower area. Both areas are thus well adapted for graphing physical data, either in the presence of points representing chemical analyses or in either area when the chemical analyses for that area are not available or are not used.

Need for Standard Practice in Graphic Presentation

Graphic methods lose much of their value and may even become confusing unless certain conventional methods are followed. One of the most important rules in ordinary practice is that scales should increase upwards or to the right from the zero point, whenever possible. This conventional presentation is a part of the conscious or unconscious education of every technical man, and should never be violated without good reason, any more than words should be spelled backwards or upside down.

The multibasic coal chart has been constructed in accordance with this principle, regardless of the orientation of previously published coal

classification charts. That there is a need for uniformity is very evident. For example, the ordinary rectangular diagram has often been used in coal classification work but each of the four corners has been appropriated by some investigator as the starting point of the coal series! Such conditions lead to needless confusion, and delay the general acceptance and understanding of coal classification methods.

It will be noted from Fig. 1 that in the upper area of the multibasic coal chart, the volatile-matter scale increases upwards in the conventional way, and that consequently the fixed-carbon scale is reversed. These scales necessarily read in opposite directions, and the volatile-matter scale was made to read upwards in order to secure the best possible correlation with ultimate analysis data.

On the condensed form (Fig. 2) of the multibasic coal chart (which is used to save space in Figs. 3, 4, 5 and 6 accompanying this paper) only the reversed fixed-carbon scale is shown. However, for many purposes, coal technologists will wish to use the complete multibasic coal chart (Fig. 1) on which volatile matter on any purity basis can be shown, and for which the scale reads in the normal direction.

Simplicity of Construction of Multibasic Coal Charts

The inclination of B.t.u. lines, relative position of upper and lower areas and the proportions of the various scales of the multibasic coal chart, were determined empirically after studying hundreds of coal analyses, calculated to various purity bases. Fortunately, it has been possible to represent the complex relationships by a relatively simple construction.

For example, the vertical distance representing 1 per cent. hydrogen is exactly equal to the distance representing 10 per cent. volatile matter or fixed carbon in the upper area, while the distance representing 1 per cent. carbon is exactly equal to a horizontal distance of 200 B.t.u. The B.t.u. lines have an inclination of 2:3 and from Figs. 1 and 2 it is evident that the B.t.u. value at any point on the top of the chart (0 per cent. fixed carbon line) is exactly 200 times the percentage of carbon directly beneath it in the lower area.

As a result of these simple relationships, it is possible to construct a multibasic coal chart (or any desired portion of the chart) quickly and easily on any piece of ordinary cross-section paper. All that is necessary is to draw in the sloping B.t.u. lines and to number the various scales according to the proportions just described, using Fig. 1 or Fig. 2 as an example.

Comparison with Other Graphic Systems

The general relations existing between ultimate analysis, proximate analysis and B.t.u. have not only been made use of in the Dulong and the

Goutal formula and indicated in Gruner's tabulation, but have been presented graphically by Ralston[9] and by Seyler.[10]

Ralston, in his classic paper, used analyses calculated free of moisture, ash, sulfur and nitrogen, and showed that when coal analyses are graphed on a triaxial diagram (similar to the lower area of the multibasic coal chart) it was possible to draw isovolatile lines and isocalorific lines with a fair degree of definiteness. Neither series of lines was uniformly spaced or parallel. From his discussion and graphs, it appears that ultimate analysis determines volatile matter within about ± 5 per cent. and calorific value within ± 250 to ± 1000 B.t.u. (depending upon the rank of coal). The writer has independently found a similar range of variation. The Ralston diagram is a most important generalization, but it cannot be successfully used for graphing B.t.u. and volatile matter (or fixed carbon) values, since the degree of correlation is not sufficiently good for this purpose.

To get around this limitation the multibasic coal chart was developed, having two adjoining areas on which B.t.u. and proximate analysis data as well as ultimate analyses could be graphed with equal precision, the degree of correlation being shown by the relative position of the points in the respective areas. The upper area is equivalent to that used by Parr,[11] Thom,[12] and Stansfield and Sutherland,[13] but the rectangular field used by these investigators has been converted into the less usual rhomb, to inprove correlation with the lower, or ultimate analysis area.

When it is not desired to obtain graphic correlation with ultimate analysis, the upper area may be converted to the ordinary rectangular field (B.t.u. lines vertical instead of slanting) and used alone without losing some of the advantages that have been described.

Availability of Charts

In order to facilitate the work of the Coal Classification Committees, several forms of blank multibasic coal charts have been prepared and are ready for distribution. These charts consist of close-ruled, blue-line prints, on good quality white paper, and are folded to 8½ by 11 in. (letter size).

---

[9] O. C. Ralston: Graphic Study of Ultimate Analyses of Coals. U. S. Bur. Mines *Tech. Paper* 93 (1915).

[10] C. A. Seyler: The Classification of Coal. *Trans.* A. I. M. E. (1928) **76,** 189.

[11] S. W. Parr: The Classification of Coal. Univ. Ill. Eng. Expt. Sta. *Bull.* **180** (1928).

S. W. Parr: The Classification of Coal. *Jnl. Ind. & Eng. Chem.* (1922) **14,** 919.

[12] W. T. Thom: Moisture as a Component of the Volatile Matter of Coal. *Trans.* A. I. M. E. (1925) **71,** 282.

[13] E. Stansfield and J. W. Sutherland: The Classification of Canadian Coals. Can. Min. & Met. *Bull.* (1929.)

A limited additional supply of these charts has been prepared from which copies will be distributed without charge or obligation to those who wish to use them for the systematic study of coal analyses and physical test data. The nature of the individual charts is described below. Charts 10 and 11 are the most general, and are particularly easy to use and understand. The other charts consist of a portion of the multibasic coal chart on a larger scale (No. 6) or charts in which the vertical scale has been exaggerated two to five times (Nos. 8 and 9) to permit a wider spacing of points, or charts in which the user selects and writes in his own scales (Nos. 7 and 9).

Blank Multibasic Coal Charts

Available for limited distribution:

No. 6. Lignite to anthracite (10 divisions per inch) 1 in. = 1 per cent. C, 1 per cent. H, 10 per cent. fixed carbon and 200 B.t.u. This chart consists of the area in Fig. 2 lying between 32 per cent. and 94 per cent. carbon, and 2.5 per cent. hydrogen to 25 per cent. fixed carbon.

No. 7. Dual-purpose work sheet (10 divisions per centimeter) for studying B.t.u. and proximate analysis, ultimate analysis, or physical data on any scale selected by user. For example, 1 per cent. carbon = 1, 2, 5 or 10 cm., as desired. This chart is ruled with B.t.u. lines having the normal slope of 2:3.

No. 8. Chart similar to Fig. 2, but with vertical scales expanded two times (10 divisions per centimeter). 1 cm. = 1 per cent. C, 0.5 per cent. H, 5 per cent. fixed carbon and 200 B.t.u.

No. 9. Dual-purpose work sheet similar to Chart No. 7 except that the vertical scale has been expanded five times. This chart is ruled with B.t.u. lines having a slope of 10:3.

No. 10. Chart similar to Fig. 2, but with additional ruled space for graphing physical data, etc., below ultimate analysis area (10 divisions per half-inch). ½ in. = 1 per cent. C, 1 per cent. H, 10 per cent. fixed carbon and 200 B.t.u.

No. 11. Complete chart similar to Fig. 1 (10 divisions per half-inch). ½ in. = 1 per cent. C, 1 per cent. H, 10 per cent. fixed carbon or volatile matter and 200 B.t.u.

These charts should be sufficient for almost any need that will arise. The advantages of this form of graphic presentation are so numerous that it is hoped that fuel technologists will become familiar with multibasic coal charts and use them extensively.

Acknowledgment

The author gratefully acknowledges the assistance of Mr. J. J. S. Sebastian in preparing the charts described in this paper.

DISCUSSION

[The following papers were presented in succession at the February, 1930, meeting and were discussed jointly: "Coal Classification; a Review and Forecast," by George H. Ashley; "Outline of a Suggested Classification of Coals," by David White; "Status of Coal Classification in Canada," by R. E. Gilmore, and "The Multibasic Coal Chart," by H. J. Rose. The oral discussion follows.]

D. B. Reger, Morgantown, W. Va., stated that the time may come when such classifications will be a matter of historical rather than of practical interest. Since the composition of coal is largely fortuitous rather than in accord with a natural law, why not recognize this fact and instead of attempting to base classification on the composition of coal, go to the user, find out his needs and classify our coals accordingly. Such a plan would reduce the classification into a few groups, such as domestic, steam smithing, gas coals, etc. These features are basic to the public which cares little about the composition of a coal. Indeed, we are now reaching a point where we are able to transform our raw material into the product desired; *e. g.*, a synthetic anthracite coal can be made which will compete with natural anthracite.

It seems characteristic of the coal industry that it fails to consider the desires of the consumer, an attitude once held by the cement industry in making natural cement but which it long since abandoned and now prepares its products to meet such specifications as are laid down for its products by the consumer. And so it is with pigments and many other industrial and mineral products.

If the operators of one region can produce an ideal fuel of a given type without processing, let them do so, but let operators of other regions adapt their processes to the production of a fuel to meet the consumer's needs and let him pay accordingly.

Dr. Ashley responded that three committees are now at work on coal classification; one to prepare a use classification, one a marketing classification and the third a scientific classification based, presumably, on the composition. One objection to a strictly use classification is that the limits of usage are variable and poorly defined, both as to time and place. For example, Indiana coals which 20 years ago were regarded as noncoking are now being coked, and coke-oven operators in Colorado are coking coals which would not be considered for that purpose in the East. Furthermore, household coal includes all kinds, from lignite to anthracite.

On the other hand, the composition is fairly definite and consistent within reasonable limits, *e. g.*, the B.t.u. value increases with the fixed carbon as a rule.

It is gratifying to note that the differences between the classifications presented by the different authors are really not serious and are such as can be eradicated by further study and cooperation.

F. R. Wadleigh, New York, N. Y., stated that with respect to the industry's attitude in endeavoring to meet the consumer's needs, it has already adopted or proposed 170 different use classifications.

Dr. White expressed doubt as to the permanency of use classifications in view of the constantly changing uses, whereas a scientific classification would afford a permanent basis for the comparison of coals as found in nature, irrespective of any processes of treatment to which they are subjected or uses to which they are put. He congratulated Mr. Rose on the development of his methods of plotting coal analyses, predicting that they will become widely adopted because they are simple, graphic and favorable for the ready comparison of analyses. However, he thought it better to plot peat and lignite on a total-carbon rather than on a fixed-carbon basis as this would bring them into better alignment with other coals. He urged that no considera-

tions of possible political or legislative actions or requirements should be allowed to enter into the formulation of a scientific classification to impair its permanency as well as freedom from bias.

M. R. Campbell, Washington, D. C., stated in reply to Mr. Reger that there is a growing need for an accurate and widely recognized classification to which both buyer and seller can refer in case of doubt. Better business bureaus and like agencies interested in the promotion of sound trade practices now have no sure basis on which to take a stand in these matters and are continually asking for some such needed and dependable classification.

Forty years ago coal was either soft or hard, but the public is now far more discriminating and needs accurate guidance in specifying its wants.

H. L. Olin, Iowa City, Iowa, stated that in his opinion slacking properties should also be considered in such a classification as they have a profound effect on the value of a coal. Some Iowa coals which are high in moisture have slacking coefficients up to 80 per cent, while others are as low as 15 or 20 per cent.

Changes in Properties of Coking Coals Due to Moderate Oxidation during Storage

By H. J. Rose* and J. J. S. Sebastian,† Pittsburgh, Pa.

(New York Meeting, February, 1930)

When coal is stored under ordinary conditions, progressive changes take place in its chemical and physical properties. These changes are largely caused by the reaction of atmospheric oxygen with the coal substance. Although this "oxidation" or "weathering" occurs slowly in the case of high-rank coals, it is nevertheless a matter of considerable importance, since even a small amount of oxidation may affect the value of coal for carbonization purposes.

Coal storage is a necessity at all carbonizing plants. In normal times there are about 5,000,000 tons in storage at by-product coke ovens and gas retorts in the United States, as compared with an annual throughput of about 75,000,000 tons. In other words, the average coal reserve at such plants is slightly less than one month's supply. However, owing to the cost of storage and recovery, coal is usually left in stockpile for as long a time as the management thinks advisable. Storage for six to eight months is common practice and is, in fact, a necessity at certain plants which obtain their coal by water routes that are closed during winter months.

Modern by-product coke ovens operate continuously, and they are not cooled down from the time that they are put into operation until they are shut down for rebuilding—perhaps 20 years later. This continuous operation is imposed by silica-brick construction and high operating temperatures. It is therefore obvious that coke plants must at all times maintain an adequate supply of stored coal, in order to prevent a failure in supply from any cause whatever. Thus when strikes or other emergencies threaten, abnormally large coal stocks are accumulated and held at coke and gas plants.

Many of these plants have been forced, at some time or other, to use coal that has been stored for several years. Such coal may fail to coke at all, or it may produce a dirty appearing, weak coke of reduced value. The yields of salable by-products may also be distinctly lowered. However, it must not be assumed that slight oxidation or heat effects are always injurious. Certain high-volatile coals produce a stronger, blockier

* Director, General Laboratory Department, The Koppers Research Corpn.
† Research Chemist, Coal and Coke Division, The Koppers Research Corpn.

coke after they have been slightly weathered. It is thus apparent that coking coals vary not only in their ability to withstand oxidation, but also in the effect which moderate oxidation produces.

For these reasons it is apparent that reliable information is needed at by-product coke plants regarding the behavior of different coals when stored. Practical experience is perhaps the best teacher, but many puzzling situations will be encountered in practice. This is because such factors as the method of storage, size and location of stockpiles, segregation of sizes, moisture content of coal, etc., have a combined effect which often outweighs inherent differences in the coal itself. In fact, small differences in the above factors may have such a great effect that a coal which is stored successfully at one plant sometimes gives considerable trouble at a neighboring plant where storage conditions appear to be only slightly different.

While these variables greatly complicate the situation, the fact remains that it is desirable, nevertheless, to have reliable information on the inherent differences in the behavior of various coals when oxidized. Characteristic differences of this sort can best be studied by laboratory tests. Such information would have practical value, because in the United States many coke plants have a wide choice of coals and they attempt to select those coals which, all things considered, best fit their needs. In some cases, a knowledge of the stocking properties of a coal is just as important as a knowledge of its ash or sulfur content.

The need exists also for a method whereby a works chemist can test stored coal and determine the degree of weathering with certainty, even when the total change in composition is small.

In another paper[1] we have discussed and graphically presented the changes which occur when different types of bituminous coal are subjected to various conditions of oxidation and heat. Under fairly severe conditions, the proximate analysis, ultimate analysis and calorific value of coal are considerably altered, and the nature and degree of changes in analysis give an indication of the conditions to which the coal has been subjected.

However, it has been generally believed that changes may take place in coal which affect its value for carbonization purposes, and yet do not cause a sufficient change in composition to be evaluated, or even detected with certainty, by ordinary methods of analysis. The object of the present paper is to examine the value of ordinary coal analyses for this purpose, and to report some preliminary results of an investigation designed to establish a more sensitive laboratory method for measuring the effects of moderate oxidation.

---

[1] H. J. Rose and J. J. S. Sebastian: Coal Storage: Effect of Oxidation and Heat on the Analysis of Bituminous Coal. Presented before Gas and Fuel Division, Amer. Chem. Soc., Minneapolis Meeting, September, 1929.

Many papers have been published on the effect of oxidation and heat on bituminous coals. The data which they contain are of limited value for our present purpose for various reasons, two of which are as follows:

1. In some cases data are given on the analysis of coals before and after stockpile weathering. Such samples have been subjected to continually varying and usually unknown conditions of temperature and oxygen supply, and it is difficult to use the analytical results in any systematic study.

2. In many laboratory investigations, coals have been subjected to controlled but highly accelerated conditions. By the use of high temperatures, an attempt has often been made to reproduce in a few hours or days conditions which require months to develop in ordinary storage. Furthermore, such tests have often been made in pure dry oxygen rather than in air. It is not always a simple matter to prove whether or not highly accelerated tests give the facts that are desired for practical use.

Nature of Coal Oxidation Reactions

The following principles should be clearly understood before any attempt is made to study the analyses of oxidized coal. As a result of numerous published investigations, it is now fairly well established that, within the temperature range under consideration, the oxidation process may be considered as occurring in two stages.

In the first stage, coal absorbs oxygen, presumably forming an unstable, solid, coal-oxygen complex. During this absorption process, the sample gains in weight, and a part (perhaps one-third),[2] of the total oxidation heat is evolved at this time. Since this reaction is not accompanied by a significant evolution of gases, the principal change in the percentage composition of the oxidized coal is due simply to its "dilution" with oxygen.

The second stage of the reaction is characterized by the breakdown of the coal-oxygen complex and the evolution of carbon dioxide, water and some carbon monoxide, together with the remainder (*i. e.*, two-thirds) of the total heat of oxidation. Owing to these changes and the decided loss in weight, the percentage composition of the coal is again altered, but in a different direction. Since the first and second stages of the oxidation reaction thus have an opposite effect on the percentage composition, they produce some curious and, at first sight, anomalous changes in analysis. We have discussed and explained these changes in more detail in another paper.[3]

---

[2] J. D. Davis and J. F. Byrne: Spontaneous Combustion of Coal. Characteristics Shown by an Adiabatic Calorimeter. *Ind. & Eng. Chem.* (1925) **17**, 125.

[3] H. J. Rose and J. J. S. Sebastian: *Op. cit.*[1]

It will be realized, of course, that the above two reactions usually proceed simultaneously. At low temperatures the oxidation rate is slow and the absorption reaction predominates, although gaseous products are slowly evolved. As the temperature rises, the oxidation rate increases rapidly and both oxygen-absorption and gas-evolution reactions become active. The rate of gas evolution accelerates at about 80° C. and becomes rapid above 100° C.[4,5,6,7] This decomposition reaction continues to accelerate with rising temperature until a condition of dynamic equilibrium is reached, when the weight of oxygen liberated in gaseous products per unit time is equal to the weight of oxygen which reacts with the coal in the same time.

From the above discussion it is clear that when coals are oxidized for the purpose of studying analysis changes it is important to conduct the oxidations under conditions which will give the desired balance between oxygen absorption and breakdown of the coal-oxygen complex. If one temperature only is to be selected for laboratory experiments, what should that temperature be?

In actual practice, stockpile temperatures vary from subzero conditions at the surface of a pile in winter to the high temperatures that are encountered in the interior of a pile that is about to catch fire. However, some plants consider 70° to 80° C. as the upper limit for safety in practical coal storage. That is, when temperatures reach or exceed 80° C. (176° F.), the coal is moved promptly before it rapidly rises to the ignition point. Other plants move stored coal before it heats to this temperature.

After a consideration of various theoretical and practical aspects, a temperature of 80° C. was selected for our work, and the results indicate that this was a satisfactory temperature for our purpose.

Experimental Method

In developing an apparatus for the slow oxidation of coal under controlled conditions, such factors as size of coal particles, temperature, oxygen concentration and duration of test are all of obvious importance. We desired a test method which would control these conditions and enable us to destroy the coking properties of a good coking coal not after a few hours or days, but after weeks or months of continuous oxidation, as is the case when coal is stored.

---

[4] J. D. Davis and J. F. Byrne: *Op. cit.*

[5] W. A. Bone: Coal and Its Scientific Uses, 158. London, 1918. Longmans, Green & Co.

[6] S. W. Parr and R. T. Milner: The Oxidation of Coal at Storage Temperatures. *Ind. & Eng. Chem.* (1925) **17**, 115.

[7] H. C. Porter and O. C. Ralston: A Study of the Oxidation of Coal. U. S. Bur. Mines *Tech. Paper* 65.

A detailed description of our apparatus and experimental method is given at the end of this paper, but may be briefly summarized here, as follows:

Freshly mined samples of coking coal were pulverized to pass a 100-mesh screen and 50-gram portions were at once placed in glass U-tubes, which were then weighed and immersed in a constant-temperature bath maintained at 80° C. (176° F.). Moist air, preheated to 80° C., was continuously passed through the coal at the rate of 1 liter (0. 035 cu. ft.) per hour. This rate did not cause an appreciable rise in temperature of the coal, and the oxygen percentage at the outlet was substantially the same as in normal air. Six or more tubes of each coal were subjected to oxidation simultaneously. At appropriate intervals the U-tubes were successively withdrawn, and their contents tested.

When three good coking coals of different rank were oxidized under these conditions, their coking properties were much reduced, but not entirely destroyed, after one to two months of continuous oxidation. In other words, the time factor in this laboratory test approached that encountered in practice.

Coals Tested

Three distinctly different types of coking coal were selected for this investigation. These coals were a 17 per cent. volatile-matter coal from Pocahontas No. 3 seam and a 32 per cent. volatile-matter coal from Powellton seam, West Virginia, and a 37 per cent. volatile-matter coal from Elkhorn No. 3 seam, eastern Kentucky.

These coals have a well established reputation and are used for by-product coking throughout the entire northeastern quarter of the United States, where more than 80 per cent. of the coke is produced.

Pocahontas Coal.—Pocahontas coal is a strongly coking coal which is never coked alone in by-product ovens, on account of its expanding properties. Instead it is mixed with high-volatile coals in various proportions up to 60 per cent. (but usually 10 to 30 per cent.) to increase the size and strength of the coke. Coal-carbonizing plants usually purchase slack sizes (screenings) of Pocahontas coal, because the fines from West Virginia low-volatile coals usually contain less ash and are cheaper than the larger sizes. Under some conditions of storage, low-volatile slack coal heats rather easily and stockpile fires have often been encountered. This is in spite of the fact that such coals have a higher ignition temperature and are inherently less reactive with oxygen than most high-volatile coking coals. This is a case where differences in practical storage conditions outweigh the inherent differences between the coals. For while American coke plants usually use slack sizes of low-volatile coking coals, they obtain their high-volatile coal supply in run-of-mine or larger sizes. This fact helps to explain why inherently less active low-volatile coal may require

more attention in storage than many high-volatile coals. It is usually difficult to correlate closely laboratory oxidation results on dry powdered coal with practical coal-storage experience, because of the complexity of conditions which exist in a large stockpile and the lack of accurate information regarding these conditions.

Powellton Coal.—The area in West Virginia which is underlain by the Powellton seam is small, yet this coal has been produced in rather large quantities for many years, and it is in demand at many by-product gas and coke plants. It is coked sucessfully both alone and in mixture with other coals. Powellton seam coal is rather friable and the run-of-mine size contains few large lumps. However, it enjoys an excellent reputation for its ability to stand storage without deterioration of coking properties and it has shown little tendency to heat in storage.

Elkhorn Coal.—Elkhorn coal has long been used as a gas coal, and for making metallurgical coke when mixed with low-volatile coals. It is a strong and blocky coal, and the run-of-mine product has a large proportion of big lumps. Elkhorn coal contains more oxygen than the two types mentioned above and it loses its coking properties more rapidly when oxidized under exactly the same conditions. However, owing to the preponderance of large lumps in the run-of-mine product, Elkhorn coal does not give the trouble in storage that might be anticipated from laboratory tests.

Analysis of Coals As Received

Following are the analyses of the three coals as received. (For analyses on the dry, ash-free basis, see Tables 3, 5 and 7.)

TABLE 1.—*Analyses of Freshly Mined Coal, As-received Basis*

Coal seam	Pocahontas No. 3	Powellton	Elkhorn No. 3
County	McDowell	Fayette	Eastern Kentucky
State	W. Va.	W. Va.	Kentucky
Description of sample	Slack	Run-of-mine	Run-of-mine
Proximate Analysis (Per Cent.)			
Moisture	0.6	0.9	1.4
Volatile matter	16.7	32.0	37.0
Fixed carbon	75.7	61.7	58.8
Ash	7.0	5.4	2.8
Ultimate Analysis (Per Cent.)			
Carbon	85.5	82.6	81.8
Hydrogen	4.0	4.9	5.4
Oxygen	1.8	4.8	7.8
Nitrogen	1.2	1.6	1.6
Sulfur	0.5	0.7	0.6
B.t.u. per lb. (gross)	14,500	14,450	14,310
Agglutinating value by Marshall-Bird test, grams	16,320	7,200	5,100

Tests on Fresh and Oxidized Coals

Every systematic investigation relating to the composition and properties of coal should include rather complete analysis data, otherwise its value is much reduced. Too many published papers are deficient in this respect.

In the present investigation, each sample was submitted to the following tests: (1) Proximate analysis, (2) ultimate analysis, (3) calorific value (gross B.t.u. per lb.), (4) agglutinating value (Marshall-Bird test), (5) change in weight caused by oxidation, (6) color of alkaline extract of the fresh coals and certain oxidized samples.

In determining the proximate and ultimate analyses and calorific value, the methods of the American Society for Testing Materials (Standards, 1927) were followed. The Parr oxygen-bomb calorimeter was used. Each figure in the proximate and ultimate analyses represents the average of two or more duplicate determinations.

Agglutinating values were made according to a slight modification of the Marshall-Bird[8] test. This test measures the crushing strength in grams, of carbonized buttons made from 25 g. of a mixture of 1 part 100-mesh coal with 10 parts of 40 to 50-mesh rounded-grain silica sand. Each value in the present paper represents the average crushing strength of 10 buttons. Marshall and Bird recommended that the coal samples used in this test first be dried at 105° C. for 1 hr., and then exposed to a 40 per cent. humidity atmosphere at 70° F. for 24 hr. Since we did not wish to expose our samples to oxidation at 105° C. for even 1 hr., the agglutinating tests were made either on the undried freshly mined samples or on the oxidized samples as removed from the oxidation train. With one exception the samples contained less than 0.9 per cent. moisture as tested.

The method described by Seymour[9] was used in examining the color of the alkaline extract. Five-gram portions of coal were boiled with 1:10 hydrochloric acid and filtered and washed with hot water on an asbestos filter. The coal residue was then extracted with boiling N/1 sodium hydroxide, about six washings being used to remove all of the color. The filtrate was slightly diluted to make 50 ml. and compared in Nessler tubes with standard colors prepared from solutions of inorganic salts or from strongly oxidized coals. The reference standard was an alkaline extract from Powellton coal which had been oxidized at 105° C. for a week until a coherent coke button was no longer formed in the volatile-matter determination. This extract was arbitrarily assigned a color intensity of 1000. It had a dark orange-brown color.

---

[8] S. M. Marshall and B. M. Bird: Test for Measuring the Agglutinating Power of Coals. See page 340.

[9] W. Seymour: Deterioration of Coking Property of Coal. *Blast Furnace & Steel Plant* (1921) **9**, 435.

Analytical Results

Tables 2 to 7 inclusive contain the analytical data that were obtained. Tables 2, 4 and 6 refer respectively to Pocahontas, Powellton and Elkhorn coal analyses reported on the "as tested" basis. Tables 3, 5 and 7 contain the same data, recalculated to the dry, ash-free basis.

The data on this latter basis are graphically presented by Figs. 1, 2 and 3, referring respectively to Pocahontas, Powellton and Elkhorn coal. Each of these charts presents a complete view of the simultaneous changes which occurred during oxidation. The volatile matter, carbon, hydrogen and oxygen results are the percentages (calculated to dry,

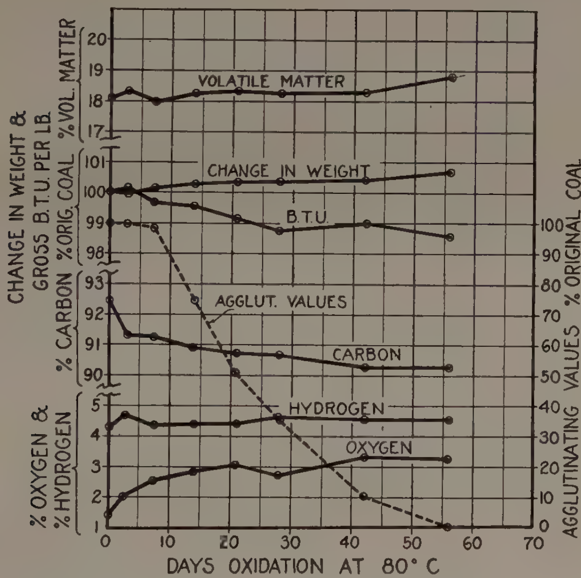


Fig. 1.—Effect of oxidation at 80° C. on Pocahontas no. 3 coal. All data except agglutinating values are calculated to dry, ash-free basis.

ash-free basis) as obtained by analysis. For convenience in comparison the change in weight, B.t.u. and agglutinating value results have also been calculated to a percentage basis on the assumption that each of these characteristics had a value of 100 per cent. in the fresh coal.

Since there is no satisfactory method for converting agglutinating values to the dry, ash-free basis, those results are shown on the "as tested" basis. The agglutinating values are read from the right-hand vertical scales in Figs. 1, 2 and 3. Owing to the great changes in this physical property which occur when coal is oxidized, the scale of agglutinating value has been compressed 10 times with respect to all other scales. This was considered necessary for the sake of convenient presentation.

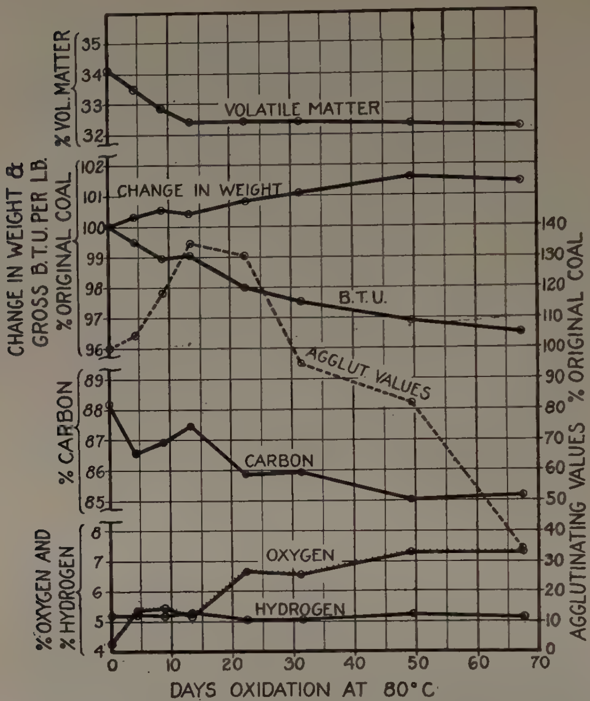


FIG. 2.—EFFECT OF OXIDATION AT 80° C. ON POWELLTON SEAM COAL. All data except agglutinating values are calculated to dry, ash-free basis.

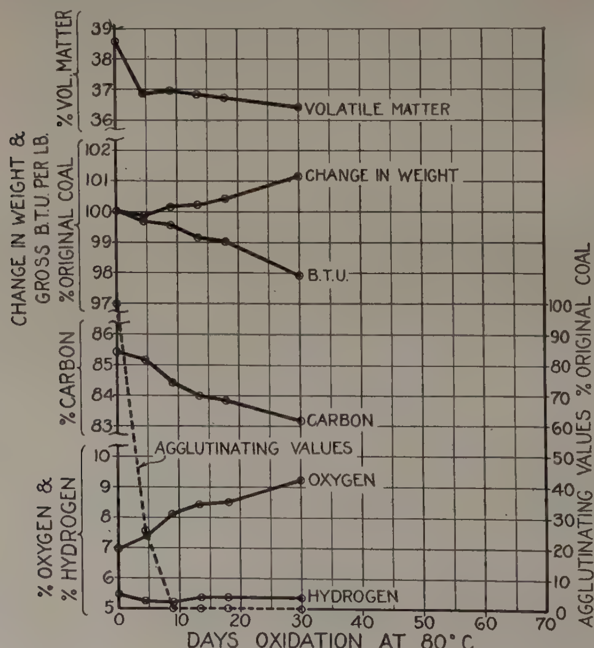


FIG. 3.—EFFECT OF OXIDATION AT 80° C. ON ELKHORN NO. 3 COAL. All data except agglutinating values are calculated to dry, ash-free basis.

TABLE 2.—*Pocahontas No. 3 Seam, McDowell County, West Virginia*

Analyses are Shown on the "As Tested" Basis and Refer to Raw Coal, or Samples as Removed from Oxidation Apparatus

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.						
		3	7½	14	21	28	42	56
Proximate Analysis (Per Cent.)								
Moisture	0.6	0.2	0.2	0.2	0.2	0.2	0.2	0.1
Volatile matter	16.7	17.0	16.7	17.0	17.0	17.0	17.0	17.6
Fixed carbon	75.7	76.0	76.2	76.1	75.9	76.1	76.0	75.5
Ash	7.0	6.8	6.9	6.7	6.9	6.7	6.8	6.8
Ultimate Analysis (Per Cent.)								
Carbon	85.5	85.0	84.7	84.5	84.2	84.4	84.0	84.1
Hydrogen	4.0	4.3	4.1	4.1	4.1	4.3	4.2	4.2
Oxygen	1.8	2.1	2.6	2.9	3.1	2.7	3.2	3.1
Nitrogen	1.2	1.2*	1.2*	1.2*	1.2*	1.3*	1.3*	1.3
Sulfur	0.5	0.6	0.5	0.6	0.5	0.6	0.5	0.5
B.t.u. (gross) per lb	14,500	14,620	14,520	14,530	14,440	14,420	14,450	14,400
Weight per 100 g. original sample	100	99.32	99.70	99.66	99.87	99.70	99.74	99.99
Agglutinating value (Marshall-Bird test)	16,320	16,310	16,010	12,160	8,230	5,820	1,690	Nil
Color intensity of alkaline extract	4.3				5.2			6.3

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

TABLE 3.—*Pocahontas No. 3 Seam, McDowell County, West Virginia*

Analyses Calculated to the Dry, Ash-free Basis

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.						
		3	7½	14	21	28	42	56
Proximate Analysis (Per Cent.)								
Volatile matter	18.1	18.3	18.0	18.2	18.3	18.3	18.3	18.9
Fixed carbon	81.9	81.7	82.0	81.8	81.7	81.7	81.7	81.1
Ultimate Analysis (Per Cent.)								
Carbon	92.4	91.4	91.3	90.9	90.7	90.7	90.2	90.3
Hydrogen	4.3	4.7	4.3	4.4	4.4	4.6	4.5	4.5
Oxygen	1.4	2.0	2.5	2.8	3.0	2.7	3.3	3.2
Nitrogen	1.3	1.3*	1.3*	1.3*	1.3*	1.4*	1.4*	1.4
Sulfur	0.6	0.6	0.6	0.6	0.6	0.6	0.6	0.6
B.t.u. (gross) per lb	15,690	15,710	15,640	15,620	15,550	15,500	15,530	15,460
Weight per 100 g. of dry, ash-free original sample	100.00	99.97	100.15	100.29	100.32	100.37	100.39	100.69

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

TABLE 4.—*Powellton Seam, Fayette County, West Virginia*

Analyses are Shown on the "As Tested" Basis and Refer to Raw Coal or Samples as Removed from Oxidation Apparatus

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.						
		4½	9	13½	22½	31½	49½	67½
Proximate Analysis (Per Cent.)								
Moisture	0.9	0.1	0.5	0.3	0.5	0.3	0.3	0.5
Volatile matter	32.0	31.7	31.0	30.6	30.5	30.6	30.6	30.4
Fixed carbon	61.7	62.9	63.2	63.8	63.7	63.8	63.9	63.9
Ash	5.4	5.3	5.3	5.3	5.3	5.3	5.2	5.2
Ultimate Analysis (Per Cent.)								
Carbon	82.6	82.3	81.9	82.3	80.9	81.1	80.3	80.4
Hydrogen	4.9	5.0	4.9	5.0	4.8	4.8	5.0	4.9
Oxygen	4.8	5.1	5.6	5.1	6.7	6.5	7.2	7.3
Nitrogen	1.6	1.6*	1.6*	1.6*	1.6*	1.6*	1.5*	1.5
Sulfur	0.7	0.7	0.7	0.7	0.7	0.7	0.8	0.7
B.t.u. (gross) per lb	14,450	14,510	14,370	14,400	14,230	14,190	14,110	14,020
Weight per 100 g. original sample	100	99.40	100.06	99.77	100.36	100.43	100.80	100.83
Agglutinating value (Marshall-Bird test)	7,200	7,520	8,510	9,700	9,390	6,850	5,910	2,430
Color intensity of alkaline extract	26.4				45.5	55.6		

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

TABLE 5.—*Powellton Seam, Fayette County, West Virginia*

Analyses Calculated to the Dry, Ash-free Basis

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.						
		4½	9	13½	22½	31½	49½	67½
Proximate Analysis (Per Cent.)								
Volatile matter	34.2	33.5	32.9	32.4	32.4	32.4	32.3	32.2
Fixed carbon	65.8	66.5	67.1	67.6	67.6	67.6	67.7	67.8
Ultimate Analysis (Per Cent.)								
Carbon	88.1	86.9	86.9	87.2	85.9	85.9	85.0	85.2
Hydrogen	5.2	5.3	5.2	5.2	5.1	5.1	5.3	5.2
Oxygen	4.2	5.3	5.4	5.1	6.6	6.6	7.3	7.3
Nitrogen	1.7	1.7*	1.7*	1.7*	1.7*	1.7*	1.6*	1.6
Sulfur	0.8	0.8	0.8	0.8	0.7	0.7	0.8	0.7
B.t.u. (gross) per lb	15,410	15,330	15,250	15,260	15,100	15,030	14,930	14,870
Weight per 100 g. of dry, ash-free original sample	100.00	100.31	100.54	100.41	100.83	101.10	101.60	101.41

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

Table 6.—*Elkhorn No. 3 Seam, Eastern Kentucky*

Analyses are Shown on the "As Tested" Basis and Refer to Raw Coal or Samples as Removed from Oxidation Process

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.				
		4½	9	13½	18	30
Proximate Analysis (Per Cent.)						
Moisture	1.4	0.8	0.6	0.6	0.8	0.6
Volatile matter	37.0	35.5	35.7	35.6	35.4	35.2
Fixed carbon	58.8	61.0	60.9	61.0	61.0	61.4
Ash	2.8	2.7	2.8	2.8	2.8	2.8
Ultimate Analysis (Per Cent.)						
Carbon	81.8	82.2	81.5	81.1	80.8	80.3
Hydrogen	5.4	5.1	5.1	5.2	5.3	5.2
Oxygen	7.8	7.8	8.4	8.7	8.9	9.5
Nitrogen	1.6	1.6*	1.6*	1.6*	1.6*	1.6
Sulfur	0.6	0.6	0.6	0.6	0.6	0.6
B.t.u. (gross) per lb	14,310	14,360	14,360	14,300	14,250	14,120
Weight per 100 g. original sample	100	99.18	99.35	99.45	99.85	100.36
Agglutinating value (Marshall-Bird test)	5,100	1,290	Nil	Nil	Nil	Nil
Color intensity of alkaline extract	4.4				41.7	

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

Table 7.—*Elkhorn No. 3 Seam, Eastern Kentucky*

Analyses Calculated to the Dry, Ash-free Basis

Constituents or Properties Determined	Fresh Coal	Days of Oxidation at 80° C.				
		4½	9	13½	18	30
Proximate Analysis (Per Cent.)						
Volatile matter	38.6	36.8	36.9	36.9	36.7	36.4
Fixed carbon	61.4	63.2	63.1	63.1	63.3	63.6
Ultimate Analysis (Per Cent.)						
Carbon	85.4	85.2	84.4	84.0	83.8	83.2
Hydrogen	5.4	5.2	5.2	5.4	5.4	5.4
Oxygen	6.9	7.4	8.1	8.4	8.5	9.2
Nitrogen	1.6	1.6*	1.6*	1.6*	1.6*	1.6
Sulfur	0.7	0.6	0.7	0.6	0.7	0.6
B.t.u. (gross) per lb	14,940	14,890	14,870	14,810	14,790	14,620
Weight per 100 g. of dry, ash-free original sample	100.00	99.86	100.15	100.20	100.40	101.15

* Estimated. The percentage of nitrogen was determined only in the fresh coal and in the sample oxidized for the longest time.

Discussion of Results

Study of Relative (Percentage) Changes

Change in Weight.—Following a loss in weight during the first oxidation period (due principally to loss of moisture), all samples continued to gain in weight. The total increase after the last oxidation period amounted to 0.7 to 1.4 per cent. on the dry ash-free basis. The Pocahontas coal changed the least, while the Powellton and Elkhorn coals showed a similar gain in weight at the end of one month.

Moisture.—When examining the moisture results (Tables 2, 4 and 6), it must be kept in mind that the coals were oxidized at 80° C. in a current of air which had been substantially saturated at room temperature. The amount of moisture in equilibrium with the oxidized coal under these conditions ranged from 0.2 to 0.8 per cent. and was inversely related to the rank of the coal.

Volatile Matter.—It has been a frequent observation that the volatile-matter content of high-volatile coals is somewhat lowered by weathering. This is borne out by our results on Powellton and Elkhorn coals, which show a reduction of about 2 per cent. in volatile matter. Most of this change occurred during the early stages of oxidation. The Pocahontas coal did not show a well defined, consistent change in volatile-matter content.

Calorific Value (B.t.u.).—As was expected, all coals showed a progressive loss in calorific value upon oxidation. The total reduction in gross B.t.u. per lb., as compared with the fresh coals, ranged from 230 to 540 B.t.u. (1.5 per cent. to 3.5 per cent.) on the dry, ash-free basis. The Pocahontas coal showed the least change, while the Powellton and Elkhorn coals showed about the same loss after one month. In each case the percentage reduction in calorific value is nearly equal to the decrease in carbon and increase in oxygen percentages. However, the calorific values dropped at about twice as fast a rate as the samples gained in weight.

Carbon.—In the case of each coal the percentage of carbon progressively decreased with oxidation. As already pointed out, a fairly definite relation exists between carbon, oxygen and calorific values. The drop in carbon percentage ranged from 2.2 to 3.0 per cent., being least in the case of the Pocahontas coal.

Hydrogen.—The hydrogen percentages (dry, ash-free basis) remained practically constant, and the ultimate analysis does not indicate any appreciable loss in hydrogen during oxidation at 80° C.

Oxygen.—The progressive increase in oxygen is about equal to the percentage decrease in calorific value and carbon. The relation between oxygen content and agglutinating value will be discussed later.

Study of Analyses on the Multibasic Coal Chart

In another paper[10] we have graphically studied the changes in analysis caused by severe oxidation. Fig. 4 presents the foregoing data in a similar manner. This diagram represents a small area from the Rose multibasic coal chart.[11] This type of chart permits a direct comparison of the B.t.u.-fixed carbon (or volatile matter) relationship with the ultimate analysis. The scales for the upper and lower areas of this chart have been so selected that all similarities or differences in the progressive change in analysis become very evident.

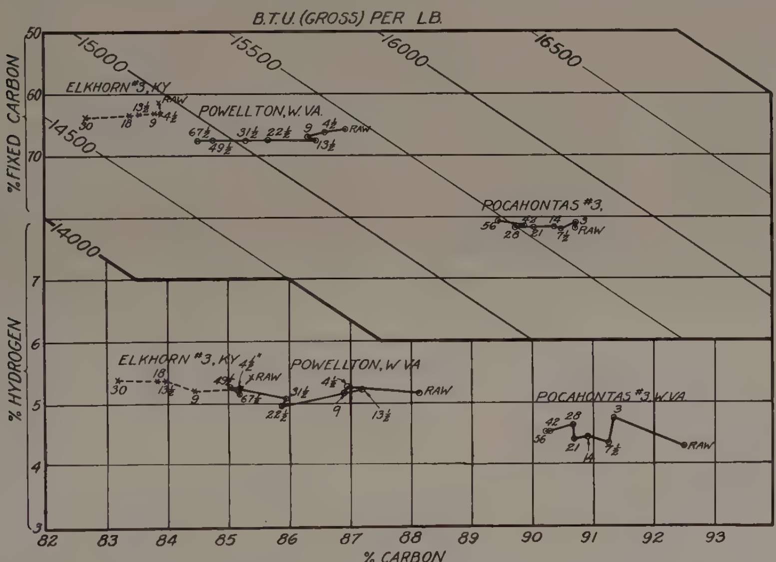


FIG. 4.—OXIDATION OF 100-MESH COAL AT 80° C.
Analyses on dry, ash-free basis after number of days indicated in figure.

Fig. 4 shows that in the case of each of the three coals tested, the course of oxidation is shown about equally well in both the upper and lower areas although all of the changes are relatively small, as compared to those caused by severe oxidation. In Fig. 4 the individual points have been connected by straight lines, instead of by drawing a smooth curve through each group of values. In the case of the B.t.u.-fixed carbon data (upper area), the analytical results appear to be very consistent and the lines connecting the values are nearly straight.

However, the lines connecting the ultimate analyses are less regular, and the question arises as to whether these irregularities are real, or simply represent difficulties in obtaining ultimate analyses of high precision.

---

[10] H. J. Rose and J. J. S. Sebastian: *Op. cit.*

[11] H. J. Rose: Multibasic Coal Charts. See page 541.

The dry oxidized samples were hygroscopic, which increased the difficulty in obtaining correct values. In spite of the care used in making these determinations and the fact that some of the variations apparently exceed the tolerances of the American Society for Testing Materials, we do not feel sure that they prove the actual existence of erratic changes in ultimate analysis due to moderate oxidation.

There is little difficulty, with modern equipment, in obtaining calorific values of excellent precision. It is our firm opinion that calorimetric determinations are of great value in the study of coal and are not reported frequently enough in papers on coal research published the world over.

Value of Analyses for Detecting a Slight Amount of Oxidation

If the typical analysis of coal from a given mine is known, is it possible to detect by analysis a slight amount of oxidation in stockpile samples?

The data in this paper show that Pocahontas coal had lost its agglutinating power (but not all of its coking property in the volatile-matter test) after 56 days' oxidation. By this time it had dropped 230 B.t.u. per lb. and 2.1 per cent. in total carbon content. After 67½ days, the Powellton coal still retained some of its agglutinating power and coking property, but had dropped 540 B.t.u. per lb. and 2.9 per cent. in carbon content. However, the Elkhorn coal had lost its agglutinating power and produced only a feeble coke button after 9 days' oxidation, at which time it had dropped about 70 B.t.u. per lb. and 1.0 per cent. in carbon content.

This shows that when Pocahontas and Powellton coals have lost most of their agglutinating power and coking property there is a corresponding change in B.t.u. per lb. and percentage of carbon which can be detected with certainty. This can hardly be said of the Elkhorn coal. Since, however, oxidation affects the practical value of coal for carbonization purposes long before the coking properties have been destroyed, it is evident that analysis results in many cases will not be sufficient definitely to show a change in the value of coal caused by moderate oxidation.

The above figures refer to the dry, ash-free basis. When weathered coals are air-dried, they retain more moisture than do fresh coals. Analyses on the air-dry, ash-free basis would help to differentiate between slightly weathered and fresh coals. Of course the air-drying would have to be carried out under strictly comparable conditions.

Study of Absolute (Actual Total) Changes

The preceding discussion referred to relative changes, which were expressed as percentages or B.t.u. per lb. Since these values were based on unit weights of samples which had already undergone changes in weight due to oxidation, they do not express the absolute or total changes of the quantities involved.

It will be instructive to calculate these actual total changes. For example, let us consider the total changes which would occur in 100 lb. of fresh Powellton coal (dry, ash-free basis) when oxidized for 67½ days under our experimental conditions. By referring to Table 5, and by multiplying each figure in the 67½-day oxidation column by 1.014 (to convert these figures to the total basis) we obtain the following comparison:

TABLE 8.—*Weight Balance of Fresh and Oxidized Coal*

	Fresh Powellton Coal, Lb.	After 67½ Days Oxidation, Lb.	Loss or Gain, Lb.
Weight of sample	100.0	101.4	+1.4
Volatile matter	34.2	32.7	−1.5
Fixed carbon	65.8	68.7	+2.9
	100.0	101.4	+1.4
Carbon	88.1	86.4	−1.7
Hydrogen	5.2	5.3	+0.1
Oxygen	4.2	7.4	+3.2
Nitrogen	1.7	1.6	−0.1
Sulfur	0.8	0.7	−0.1
	100.0	101.4	+1.4
B t.u. (total)	15,410,000	15,080,000	−33,000

When the total losses or gains for each coal are calculated in this way, we obtain the following comparison:

TABLE 9.—*Absolute Losses or Gains during Oxidation in Lb. per* 100 *Lb. Original Coal (Dry, Ash-free Basis)*

	Pocahontas	Powellton	Elkhorn	Pocahontas	Powellton
Days oxidized at 80° C	56	67½	30*	28*	31½*
Weight of sample	+0.7	+1.4	+1.2	+0.4	+1.1
Volatile matter	+0.9	−1.5	−1.8	+0.3	−1.4
Fixed carbon	−0.2	+2.9	+3.0	+0.1	+2.5
	+0.7	+1.4	+1.2	+0.4	+1.1
Carbon	−1.5	−1.7	−1.2	−1.3	−1.3
Hydrogen	+0.2	+0.1	+0.1	+0.3	0.0
Oxygen	+1.9	+3.2	+2.4	+1.3	+2.5
Nitrogen	+0.1	−0.1	0.0	+0.1	0.0
Sulfur	0.0	−0.1	−0.1	0.0	−0.1
	+0.7	+1.4	+1.2	+0.4	+1.1
B.t.u. loss (total)	−12,000	−33,000	−15,000	−13,000	−21,000
B.t.u. loss per lb. original sample	−120	−330	−150	−130	−210

* First month.

Since the oxidation of each coal was conducted for a different length of time the total losses or gains after about one month's oxidation are presented in the last three columns of Table 9.

The gain in weight of Elkhorn and Powellton coal after one month was about three times that of the Pocahontas coal. The same two coals lost about 1½ lb. volatile matter and gained nearly twice as much fixed carbon while the Pocahontas coal did not show a significant change in these respects. Each coal lost about 1.3 lb. total carbon per 100 lb. original sample. If this was evolved as CO_2, nearly 3.5 lb. of oxygen were required to combine with this amount of carbon.

There was no apparent loss in hydrogen, in fact there appeared to be a slight increase. If correct, this might be explained by an increase in the moisture-retaining capacity of oxidized coal.

The oxygen content of the Pocahontas coal increased 1.3 lb. after 28 days and nearly twice as much in the case of the Elkhorn and Powellton samples. Apparently, less than half of the oxygen reacting with the coal was retained by it, even at the temperature of 80° C.

The changes in nitrogen and sulfur are too small to be significant.

During the first month, the loss in British thermal units ranged from 130 to 210 B.t.u. per lb. original coal, and was greatest in the case of the Powellton coal. It is interesting to note that the actual B.t.u. loss is considerably less than the apparent loss from the ordinary analysis result. Thus Table 9 shows that the Powellton coal after 67½ days oxidation had actually lost 330 B.t.u. per lb. original sample, as compared to the 540 B.t.u. drop indicated by the ordinary analysis results in Table 5.

Agglutinating Value and Coking Property

Many persons have fallen into the error of expecting too much of agglutinating tests in predicting the practical coking behavior of coal. The agglutinating test measures the ability of a coal to cement or bind together during carbonization many times its volume of some added noncoking material. This is not the condition which exists in ordinary commercial coke manufacture. In fact we wish to point out that most of the physical conditions in the usual types of agglutinating test are at wide variance with practical coking conditions.

"Coking property" is a much abused term which is used freely but with little real discrimination. It is doubtful whether a satisfactory general definition for "coking property" exists or can even be written with our present knowledge of the physical phenomena of coking. Most existing definitions do not describe the property, but refer instead to the quality of the coke which can be obtained.

From the standpoint of the by-product coke-oven operator, a coal has satisfactory coking properties if it will produce a sufficiently large yield of strong coke of the desired size, when coked under normal conditions (which may include the admixture of other coking coals).

We have not used the Marshall-Bird agglutinating test in an attempt to predict the coking property of coals, either in the fresh condition or

after oxidation. Instead, our purpose has been to determine whether or not slight oxidation changes the agglutinating value to a much greater extent than it changes the composition and calorific value.

The results published in this paper and illustrated in Figs. 1, 2 and 3 show that in the case of the coals tested, the agglutinating values were enormously more affected by oxidation than were proximate and ultimate analyses and calorific values. (Be sure to observe that the agglutinating scale in these figures has been reduced 10 times to permit it to be shown on the same diagram with the other results.) In Figs. 1, 2 and 3 the agglutinating strength is shown as a percentage of the value of the fresh coal. For actual crushing strengths, see Tables 2, 4 and 6.

The agglutinating strength of the fresh Pocahontas coal was very high, being about 16,000 g., which is twice as great as the values for low-volatile coking coal which have previously been reported[12] using the Marshall-Bird test. However, similar high values have since been obtained by the U. S. Bureau of Mines on other samples of low-volatile coal. The agglutinating value of the Pocahontas coal was but little affected during the first week of oxidation, after which it fell at a rapid and regular rate until a zero value was reached after 56 days.

The Powellton coal, with an initial agglutinating strength of 7200 g., showed a most interesting behavior. When oxidized at 80° C. the agglutinating value increased by 2500 g. (35 per cent.) and it was not until a month had elapsed that the value had again dropped as low as that of the original fresh coal. After 67½ days' oxidation the agglutinating value still was one-third that of the fresh coal.

The Elkhorn coal started with an agglutinating value of 5100 g. which dropped to zero within nine days of oxidation.[13]

Coke Buttons from Volatile-matter Test

We have noted a very interesting relation which is shown by Figs. 6, 7 and 8. These figures show that in the case of the three coals tested, the height of the volatile-matter coke buttons varies directly with the agglutinating values of the coal, regardless of whether the values are increasing or decreasing. Furthermore, the volatile-matter coke buttons had assumed a low and fairly characteristic form by the time that the agglutinating value had been reduced to about 2000 g. The coals, however, continued to form a fair quality volatile-matter coke button after they were no longer able to cement together 10 times their weight of sand. This is not surprising. It must also be remembered that the very rapid heating in the volatile-matter test favors the formation of coke from

[12] S. M. Marshall and B. M. Bird: *Op. cit.*

[13] For previous comment on ability of this coal to resist deterioration under practical conditions, see page 561.

poorly coking coal. The rate of heating is not as fast in the agglutinating test.

Relation of Agglutinating Value to Oxygen Content

The relation between agglutinating value and oxygen content of the three coals which we examined is clearly shown by Fig. 5. These results contain some elements of surprise.

During the first week, the oxygen content of Pocahontas coal increased more than 1 per cent., but the agglutinating value was practically unchanged. However, this value then dropped from 16,000 to zero while the oxygen content was increasing by only 0.7 per cent. additional (dry, ash-free basis).

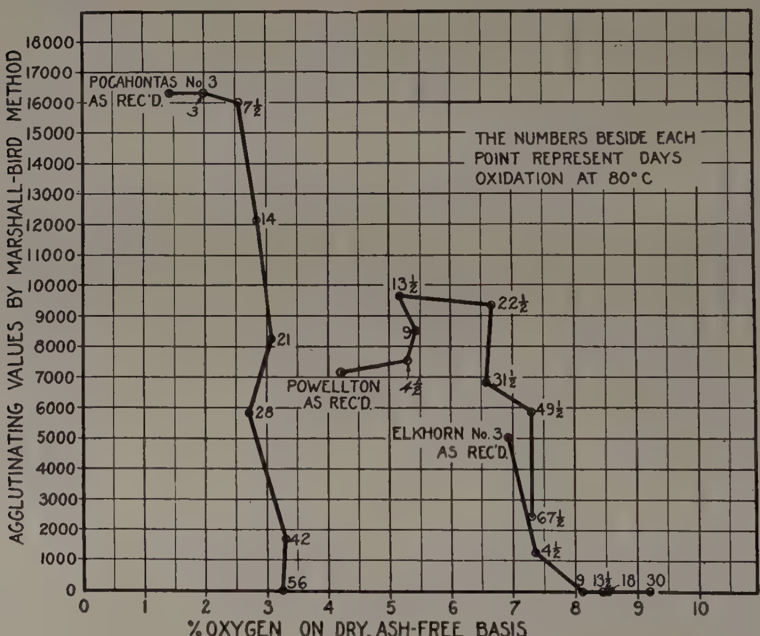


FIG. 5.—RELATIONSHIP OF AGGLUTINATING VALUE TO OXYGEN CONTENT OF COAL.

During the first three weeks the agglutinating strength of the Powellton coal increased 2200 g. at the same time that the oxygen content was also increasing rapidly (from 4.2 to 6.6 per cent.). But, when the oxygen content increased only 0.7 per cent. additional (from 6.6 to 7.3 per cent.), the agglutinating value dropped by nearly 7000 g.. This is quite different from the regular, proportional change that has been expected by some writers. It should also be pointed out that during the first month of oxidation there are two entirely different oxygen contents and two periods of oxidation corresponding to the same agglutinating value.

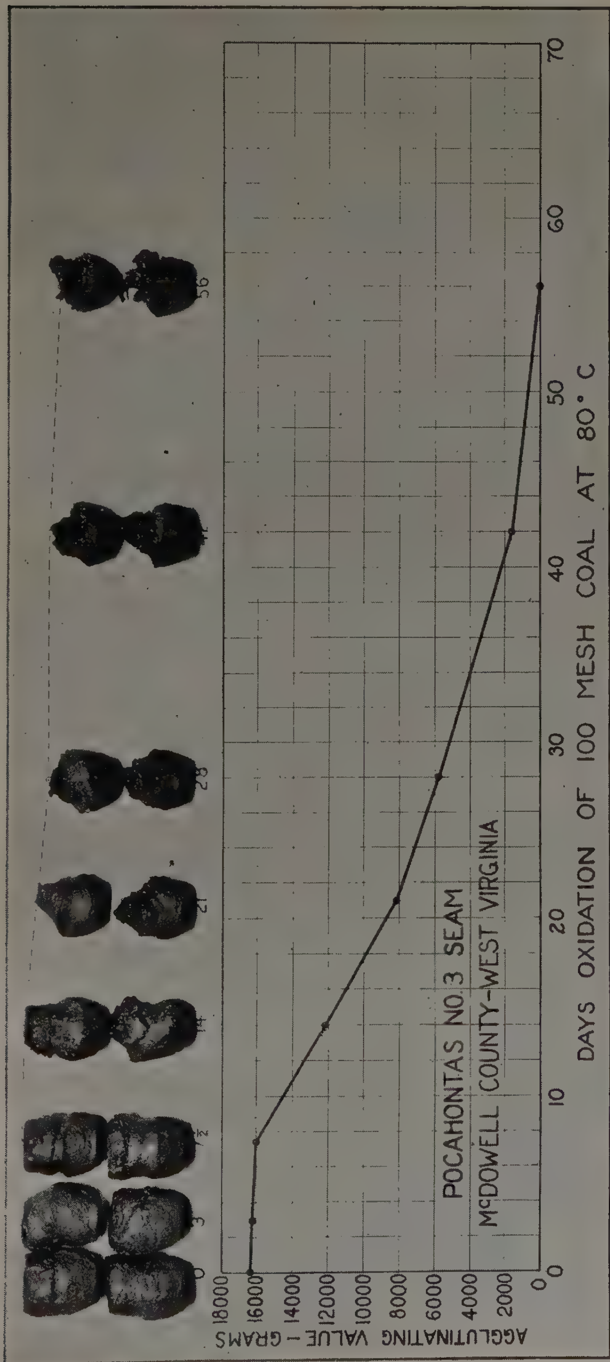


FIG. 6.—COKE BUTTONS FROM A. S. T. M. VOLATILE-MATTER TEST.
Showing relation of agglutinating value to height.

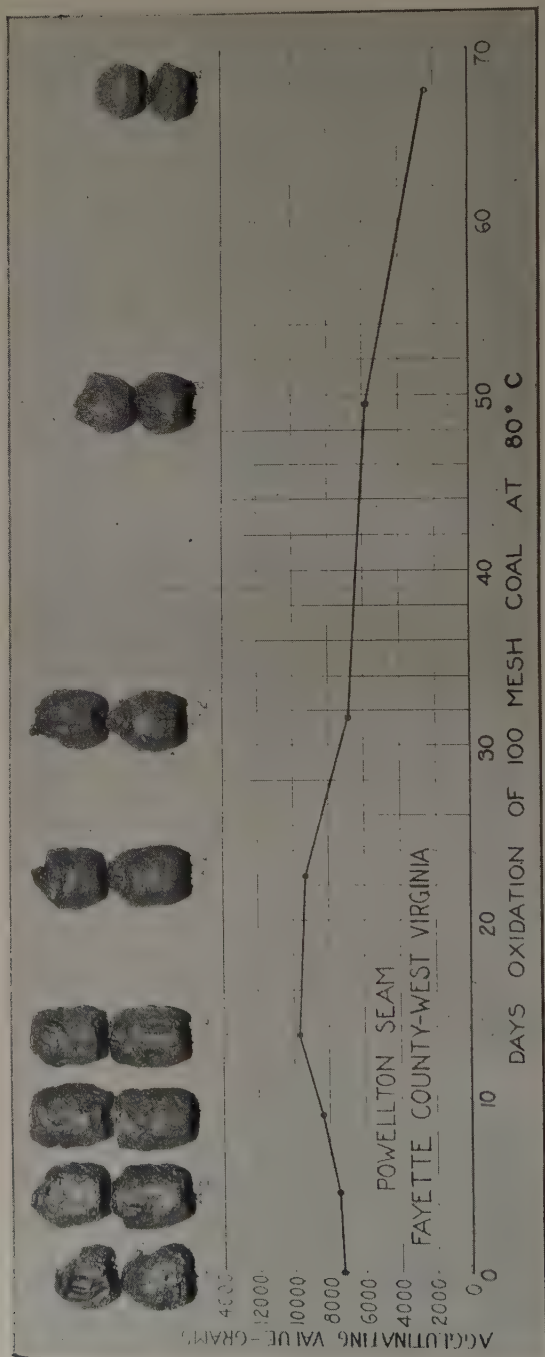


FIG. 7.—COKE BUTTONS FROM A. S. T. M. VOLATILE-MATTER TEST.
Showing relation of agglutinating value to height.

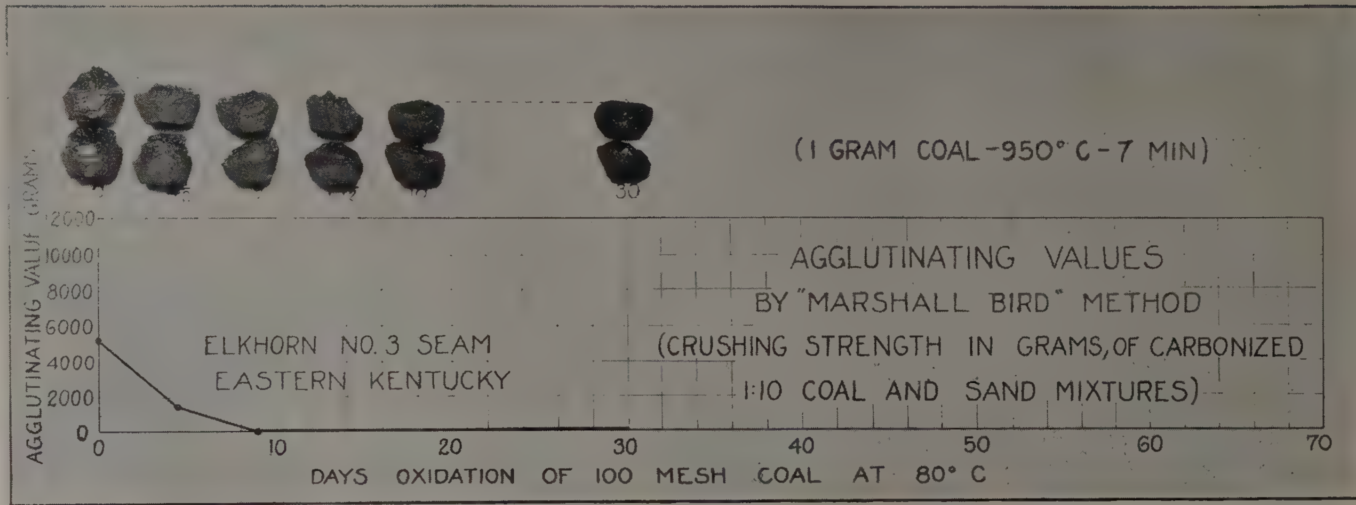


Fig. 8.—Coke buttons from A. S. T. M. volatile-matter test. Showing relation between agglutinating value and height.

The Elkhorn coal lost its agglutinating property so quickly under our experimental conditions (100-mesh coal in air at 80° C.) that only one value was obtained before this coal was unable to bind together 10 times its weight of sand. Nevertheless the Elkhorn coal continued to give a coherent coke button in the volatile-matter test when it had been oxidized for three times as long as necessary to destroy its agglutinating value as measured by the Marshall-Bird test.

The above facts indicate that (in some cases at least) there is not only a lack of direct correlation between the agglutinating value and oxygen content of oxidized coal, but that most of the increase in oxygen content occurs without much accompanying change in agglutinating value. Also, while the drops in agglutinating value take place, the oxygen content shows little change. This is an observation which merits further investigation.

We have also examined the relation between agglutinating value and the total oxygen that has reacted with the coal (this latter value can be calculated from previous data on the actual total loss of carbon by assuming that this carbon was evolved as carbon dioxide). When plotted, such data result in graphs similar to Fig. 5, but even more striking, and only serve to emphasize the lack of correlation that has just been pointed out.

In this discussion we are referring to oxidized coal samples. We have not yet seen enough data on the agglutinating values of freshly mined coals of known coking behavior to express an opinion on the degree of correlation which may be expected in such cases.

Effect of Oxidation on Color of Alkaline Extract from Coal

The most satisfactory and direct way to determine any deterioration which may have occurred during storage is to test the coal in the full-scale equipment in which it is to be used. However, such tests cannot always be conveniently made, for various reasons.

In such cases, less direct small-scale tests, or laboratory results such as chemical analyses, B.t.u. determination or agglutinating tests may be resorted to in the hope that such results can be correlated with practice. If one wishes to use indirect methods of this sort it would be worth while to find the simplest and most rapid test which could be considered satisfactory for the purpose. One test that has been proposed is the color of the alkali extract from coal.

Freshly mined coking coals contain little material that is soluble in dilute alkaline solutions. However, when such coal is oxidized, alkali-soluble substances called "humic acids" or "ulmins" are progressively formed. By a sufficiently long and severe oxidation, the coal substance can often be converted almost entirely into such products. The amount of soluble humates in the alkali extract can be estimated by titration with

an oxidizing agent,[14] or by color comparison,[15] since they usually possess a strong brown color.

It did not come within the scope of our present investigation to examine critically published methods based on the alkali-soluble material in coal, or to develop a new method. However we have made a few preliminary tests in order to make our data more comprehensive. We have already described the method that was used.

Color of Extracts from Fresh Coal

The freshly mined coal samples were pulverized for the oxidation experiments and the surplus 100-mesh coal was placed in small, tightly stoppered bottles and stored in a refrigerated chamber at a temperature close to 0° C. About four months elapsed before the bottles were opened for the alkali extraction tests.

By referring to Table 10 it is seen that the alkali extract from raw Pocahontas and Elkhorn coals had a color intensity of about 4, whereas the color of the Powellton extract was found to be six times as great. These results cannot be correlated with the rank of the coals or their agglutinating value.

Color of Extracts from Oxidized Coal

The color of the extract from Pocahontas coal was not much affected by oxidation, in fact the intensity increased only from 4 to 6 after 56 days of oxidation at 80° C. When the Pocahontas coal was oxidized at 105° C. to the total destruction of coking property (according to the volatile-matter test) the color increased only to 9.

The color intensity of the Powellton extract increased from 26 to 56 during the first month of oxidation at 80° C., but this coal gave a color intensity of 1000 when the coking property was totally destroyed by oxidation at 105° C. (about one week).

The color of the Elkhorn extract was 4 at first, and this increased to 42 after 18 days at 80° C. This coal gave an intensely dark reddish brown extract having a color intensity of 5000 when oxidized for only 36 hr. at 110° C. (with complete destruction of coking property).

In each case the color intensity of the extracts increased with oxidation, and such a method might be used to study the progress of oxidation in a given coal. However, different coals show enormous differences in the increase of color intensity of extracts, when they are oxidized under the same conditions. There does not appear to be a correlation between the color intensity of the alkaline extracts from oxidized coals and the results of the Marshall-Bird agglutinating test. Possibly other forms of alkali treatment (many of which have been proposed) might prove more useful.

---

[14] G. Charpy and G. Decorps: Sur la détermination du degré d'oxydation des charbons. *Compt. rend.* (1921) **173,** 807.

[15] W. Seymour: *Op. cit.*

TABLE 10.—*Color Intensity of Alkaline Extracts*

	Agglutinating Values	Color of Extract	Color Intensity
Pocahontas			
Raw	16,320	Very light greenish yellow	4.3
Oxidized 21 days at 80° C.	8,230	Slightly darker	5.2
Oxidized 56 days at 80° C.	0	Slightly darker	6.3
Oxidized at 105° C. to total destruction of coking property	0	Slightly darker	9.3
Powellton			
Raw	7,200	Light orange-yellow	26.4
Oxidized 22½ days at 80° C.	9,390	Same but darker	45.5
Oxidized 31½ days at 80° C.	6,850	Medium light orange	55.6
Oxidized at 105° C. to total destruction of coking property	0	Dark brown	1,000
Elkhorn			
Raw	5,100	Very light greenish yellow	4.4
Oxidized 18 days at 80° C.	0	Light orange	41.7
Oxidized 1½ days at 110° C. with total destruction of coking property	0	Very dark reddish brown	5,000

OXIDATION APPARATUS

A brief summary of the conditions under which the coals were oxidized has been given. For the benefit of those who wish to examine the method critically, or who may wish to set up apparatus for a similar purpose, the following description of the latter is offered.

Fig. 9 shows a diagrammatic elevation of the equipment and a plan showing the arrangement of apparatus in the water bath. The names and purpose of the various items, indicated by letters on these two figures, are given on page 582.

The U-tubes used to contain the coal samples were 1 in. in inside diameter and 9 in. high, and were made of Pyrex glass. The coal was poured in loosely (not compressed). There was no indication throughout the experiments of any channelling in the coal samples, and it is believed that there was a uniform contact between coal particles and the air stream. In one of the tubes, two thermometers were used, one in the inlet side and the other in the outlet side. There was no appreciable rise in temperature at the inlet side.

The rate at which the air passed through each tube was checked daily by volumetric measurement of the exit gases. About once a week the air (containing oxidation products) issuing from one of the tubes was analyzed for oxygen and carbon dioxide, using Williams pipettes. The carbon dioxide content in the air leaving the U-tubes varied from 0.0 to 0.3 per cent. The oxygen content did not fall below about 20 per cent. Usually the reduction in oxygen percentage was scarcely detectable.

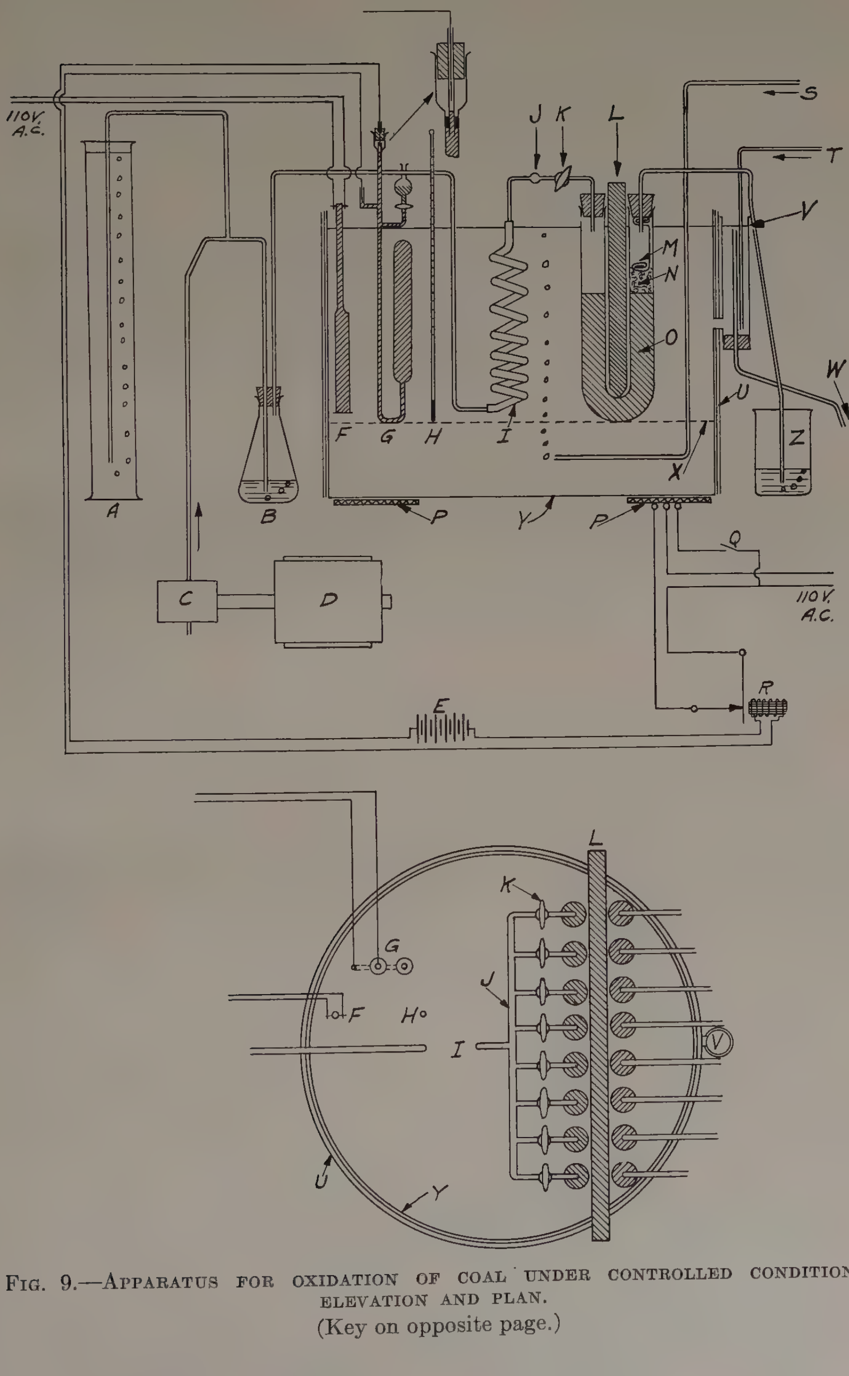


FIG. 9.—APPARATUS FOR OXIDATION OF COAL UNDER CONTROLLED CONDITIONS. ELEVATION AND PLAN.
(Key on opposite page.)

The apparatus proved to be convenient and gave controlled conditions throughout the test.

Reducing Coal Oxidation during Storage

It is not the purpose of this paper to discuss or recommend practical methods of coal storage. However, it should be pointed out that proper methods will accomplish much in reducing coal deterioration and the elimination of stockpile fires.

Excellent results are claimed for a system of storage which appears to have been used more extensively at power plants than at coke plants. In this method the stockpile is built up in layers, each of which is carefully leveled and rolled before the next layer is added. Considerable packing occurs, and the cubic-foot weight in the pile may be increased by one-third as compared to loose coal.[16] Large piles built up in this manner are well adapted to resist oxidation.

Acknowledgment

We wish to express our indebtedness to Mr. J. D. Davis of the Pittsburgh Station of the U. S. Bureau of Mines for permission to use a modified Riehle apparatus in determining the crushing strength of the carbonized agglutinating-test buttons. We also wish to thank Mr. B. Juettner of the Pittsburgh Station for assistance in making some of the agglutinating tests.

Key to Fig. 9

A. Air-pressure regulator; 20 in. water in glass cylinder.
B. Erlenmeyer flask containing water for saturating air at room temperature.
C. Centrifugal air pump.
D. Electric motor, ⅛ hp.
E. Six dry cells in series, each 1.5 volts.
F. Electric immersion heater, 500 watts.
G. Temperature regulator, mercury in glass; temperature was held at 80° C, ± 1° C.
H. Thermometer (0 to 100° C.).
I. Lead coil for preheating air.
J. Manifold to distribute air into U-tubes. (See *J* in plan.)
K. Stopcocks to regulate air supply to individual U-tubes.
L. Wooden support to hold U-tubes parallel and in place. (A perforated metal plate or piece of very heavy screen could be used for this purpose.)
M. Spiral of copper wire to prevent displacement of coal in U-tube by air pressure.
N. Cotton plug.
O. Coal sample, 50 g. just passing 100-mesh screen (0.0058-in. or 0.147-mm. square openings). Care was taken to use the minimum amount of grinding in preparing the samples.
P. Electric ring-heater with three terminals used for supplying 500 and 1000 watts.
Q. Hand-operated switch for adding 500 watts input in cool weather.
R. Relay operated by regulator *G*, controlling 500 watts input.
S. Air for stirring water bath, to make close temperature regulation possible.
T. Water supply for bath.
U. Thick asbestos-paper heat insulation for water bath.
V. Constant water-level device.
W. Excess water discharge.
X. Wire screen support.
Y. Enamelled steel water bath.
Z. Outlet for air and oxidation products.

[16] T. N. Wynne: Facts on Indiana Coal. Indianapolis, 1926. W. K. Stewart Co.

DISCUSSION

H. M. Haslam, Palmerton, Pa., inquired about the relation between the accelerated oxygenation test and natural aging. Mr. Rose replied that the effects produced by oxidizing coal at 80° C., as described in the paper, were believed to simulate satisfactorily the oxidation effects produced during actual storage.

A. C. Fieldner, Washington, D. C. (written discussion*).—Rose and Sebastian have presented clear-cut evidence of the marked effect of moderate oxidation on the agglutinating value of coal. Their paper, together with that of Marshall, Yancey and Richardson,[17] indicates that agglutinating power tests are useful in appraising a certain property of coal that is important in coking the coal and probably in the caking of coal on a fuel bed, especially with reference to the use of different types of mechanical stokers. However, we must not jump to the conclusion that this test is by itself a measure of the suitability of a coal for the manufacture of coke. It is only one of a number of factors that must be considered in this connection.

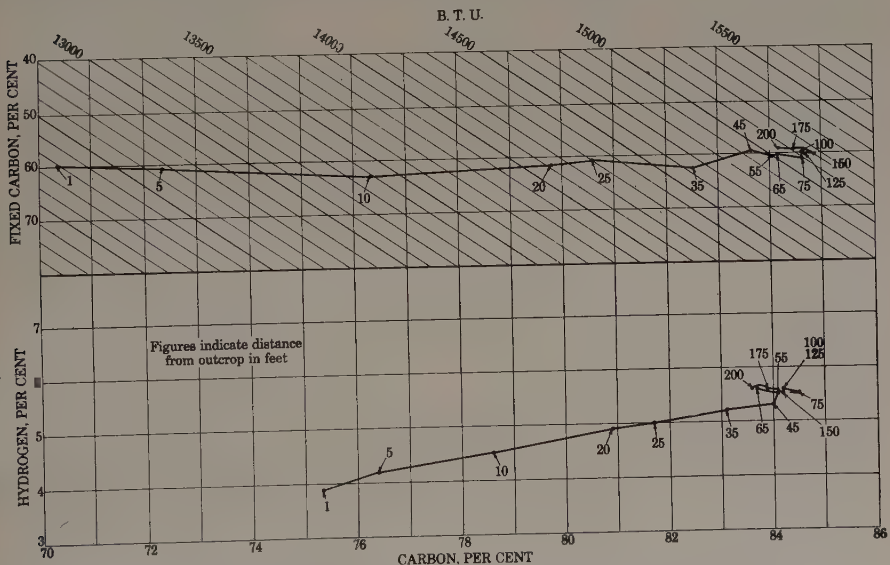


Fig. 10.—Change in composition of coal with distance from outcrop, Experimental Mine of U. S. Bureau of Mines, Pittsburgh bed. Analyses are on dry, ash-free basis.

From the data presented, it is apparent that agglutinating value tests may serve as sensitive indicators of deterioration of coking properties. It is hoped that more experimental work will be done in applying this test at gas and coke-oven plants and thus more data will be obtained on its practical application and interpretation of results.

Mr. Rose's multibasic coal chart is an excellent graphic means of showing relationships between the proximate and ultimate analysis of coal and other properties. For example, Fig. 10 shows the change in composition of the Pittsburgh coal bed at the

* Published by permission of the Director, U. S. Bureau of Mines.

[17] S. M. Marshall, H. F. Yancey and A. C. Richardson: Loss in Agglutinating Power of Coal Due to Exposure. See page 389.

U. S. Bureau of Mines Experimental Mine, with distance from the outcrop. These data are taken from *Technical Paper* 35.[18] Beginning at 1 ft. from the outcrop the coal shows a progressive increase in percentage of carbon and hydrogen (and consequent decrease of oxygen) and increase in heating value, until at approximately 60 ft. from the outcrop the composition becomes constant; that is, the cover is great enough to prevent oxidation and deterioration of the coal. The analytical values are given on a moisture and ash-free basis. The chart has the advantage of showing the essential factors of the proximate and ultimate analyses and the British thermal units in close juxtaposition on a compact graph. I regret that we did not determine agglutinating values when this work was done. A comparison of the results obtained from these coals, which were very slowly oxidized over long periods of time, with those of the authors on accelerated oxidation tests at higher temperatures would have been interesting.

---

[18] H. C. Porter and A. C. Fieldner: Weathering of the Pittsburgh Coal Bed at the Experimental Mine near Bruceton, Pa. U. S. Bur. Mines *Tech. Paper* 35 (1914).

Review of Methods Used in Coal Analysis, with Particular Reference to Classification of Coal*

By A. C. Fieldner,† Washington, D. C.

(New York Meeting, February, 1930)

The usual analytical determinations made in analyzing coal are comprised in the proximate and ultimate analysis and the determination of calorific value. The proximate analysis includes determinations of moisture, volatile matter, fixed carbon and ash; the ultimate analysis determinations of carbon, hydrogen, oxygen, nitrogen, sulfur and ash.

The proximate analysis is an empirical process, and the results, obtained, especially for volatile matter and fixed carbon, are materially influenced by the method of conducting the determinations. This fact must be kept in mind in comparing coal analyses from different laboratories for classification purposes. Unless these laboratories have followed the same standard methods, differences due to variations in the analytical methods may, in some cases, be large enough to change the classification of the coal. It is unfortunate that the determinations of volatile matter and fixed carbon, which are most used in coal classification, are the two that are subject to the greatest variations. Moisture comes next in the scale of variation.

The ultimate analysis is less influenced by method, since it comprises the determination of definite chemical elements. However, even the ultimate analysis is subject to errors, due to inorganic carbon from carbonates and to inorganic hydrogen from water. For these reasons it is evident that the Committee on Coal Classification must be informed on the evolution of methods of coal analysis in America and the average differences in analytical results that may be expected from variations in methods. The committee should be informed also on which methods should be accepted as standard for classifying coal and on the magnitude of the differences in results that may be expected from duplicate samples of the same coal sent to different laboratories for analysis by the same standard methods.

It is the purpose of the present paper to provide this information. The subject will be discussed under the following main heads:

---

* Revision of Report No. 1 (on analytical methods) of Subcommittee on Origin, Composition and Properties of Coal, and Methods for Their Determination. Published by permission of the Director, U. S. Bureau of Mines.

† Chief Chemist, U. S. Bureau of Mines, and Chairman of Sectional Committee on Classification of Coal of American Standards Association.

1. Evolution of present standard American method.
2. Influence of variations in method on analytical results.
3. Normal variations in analytical results by present American standard method.
4. Recommendations.

Evolution of Present Standard American Method

The first American standard method for analyzing coal was recommended by a committee of the American Chemical Society in 1899.[1] This report covered sampling, proximate analysis, ultimate analysis and heating effect (calorific value). The instructions on sampling, heating effect and ultimate analysis were in general terms, no specific methods being described. Detailed procedures were outlined for the proximate analysis and sulfur determination. The essentials of these methods will be given in order to trace the modifications and improvements that have been made between 1899 and 1929.

Sampling

"Crush and quarter down to size not larger than ¼-in. cube and in amount to fill a one-quart fruit jar. Seal airtight. Conduct operations rapidly to guard against moisture loss." Beyond these specific directions, the instructions are very general.

Preparation of Laboratory Sample

"Quarter down to 100 grams; grind rapidly in a mill with minimum exposure to air. Transfer part of this coarsely ground sample to an air-tight container and reserve it for special moisture determination. Grind the remainder to a powder and preserve it in a corked tube for the other determinations."

These instructions also are very general; in fact, the method outlined would not prevent material losses of moisture in sampling high-moisture coals.

Moisture

"Dry one gram of the coal in an open porcelain crucible at 104° to 107° C. for one hour, best in a double-walled bath containing toluene; cool in a desiccator and weigh covered. For accurate moisture determination of the as-received coal, use the coarsely ground sample and recalculate the analysis to this basis for as-received values."

---

[1] W. A. Noyes, Chairman; W. F. Hillebrand, C. B. Dudley. Report of the Committee on Coal Analysis to the President and Members of the American Chemical Society. *Jnl.* Amer. Chem. Soc. (1899) **21,** 1116.

Volatile Matter

"Place one gram of fresh, undried, powdered coal in a platinum crucible, weighing 20 or 30 grams, and having a tightly fitting cover. Heat over the full flame of a Bunsen burner for 7 min. The crucible should be supported on a platinum triangle with the bottom 6 to 8 cm. above the top of the burner. The flame should be 18 to 20 cm. high when burning full, and the determination should be made in a place free from drafts. The upper surface of the cover should burn clear, but the under surface should remain covered with carbon. To find 'volatile combustible matter' subtract the per cent. of moisture from the loss found here."

This method was a modification of the one given in a publication by Muck[2] and differed from the original in that the crucible was placed 6 to 8 cm. above the burner instead of 3 cm. above it, and the heating was continued a definite length of time instead of being stopped with the disappearance of flame at the edge of the lid of the crucible. The committee at that time suggested possible objections to this method because of mechanical loss from the rapid heating of noncaking coals. The committee made some experiments with noncaking coals and apparently obtained satisfactory results.

Ash

"Burn the portion of powdered coal used for the determination of moisture, at first over a very low flame, with the crucible open and inclined, till free from carbon. If properly treated, this sample can be burned much more quickly than the dense carbon left from the determination of volatile matter. It is advisable to examine the ash for unburned carbon by moistening it with alcohol."

Fixed Carbon and Sulfur

Fixed carbon "is found by subtracting the ash from the per cent. of coke."

Eschka's method was recommended for sulfur and the details for carrying out the determination were given in the report.

Ultimate Analysis

The committee deemed it unnecessary to give directions for the determination of carbon, hydrogen and nitrogen, inasmuch as these procedures are given in detail in various textbooks on organic combustion analysis.

Heating Effect, or Calorific Value

The committee recommended that the gross heating value with the water in the combustion products condensed to liquid at ordinary tem-

[2] F. Muck: Chemie der Steinkohle, 10. Leipzig, 1891. Engelmann.

peratures be reported. A bomb calorimeter was recommended but no details were given for conducting calorimetric tests.

General Use of Methods of American Chemical Society

The methods outlined came into fairly general use. They were given in the textbooks and were followed by state and national geological surveys. Perhaps the first wholesale application of the methods came with the establishment, at St. Louis, of the United States Fuel Testing Plant of the U. S. Geological Survey in 1904. A large number of coals from all parts of the United States, ranking from lignite to anthracite, were analyzed at the Fuel Testing Plant. It was soon discovered that many more details had to be standardized in the methods of the American Chemical Society in order to secure results that could be duplicated in different laboratories. Especially was this true with respect to determinations of moisture, volatile matter and fixed carbon.

The American Chemical Society method for determining moisture did not specify circulation of dry air through the moisture oven. It was found that when a moisture oven was filled with a number of crucibles containing samples of high-moisture coals, the natural air circulation in the moisture oven was quite insufficient to remove all the moisture from the samples in the specified one hour of drying. It was found necessary, therefore, to pass a current of dry air through the oven. Thoroughly dried coal was discovered to be a better desiccating agent than calcium chloride, and therefore calcium chloride desiccators were replaced by concentrated sulfuric acid. The whole procedure of sampling coal was thoroughly studied, especially with reference to unaccounted for moisture changes. As a result of these investigations, the Fuel Testing Plant and its successors, the Technologic Branch of the U. S. Geological Survey and the U. S. Bureau of Mines, adopted the general plan of shipping coal samples to the laboratory in air-tight containers, then air-drying the coal so as to reach approximate equilibrium with the air, keeping a record of the air-dry losses, then crushing the air-dry coal to 60 mesh with roll crushers and ball mills. N. W. Lord[3] and Stanton and Fieldner[4] describe modifications of the American Chemical Society methods that were used in the laboratories of the Technologic Branch of the U. S. Geological Survey and by the U. S. Bureau of Mines.

---

[3] N. W. Lord: Work of the Chemical Laboratory. U. S. Geol. Survey *Prof. Paper* 48 (1906) 174.

Experimental Work Conducted in the Chemical Laboratory of the United States Fuel Testing Plant at St. Louis, Mo., Jan. 1, 1925 to July 31, 1906. U. S. Bur. Mines *Bull.* 28 (1911).

[4] F. M. Stanton and A. C. Fieldner: Methods of Analyzing Coal and Coke. U. S. Bur. Mines *Tech. Paper* 8 (1912).

In 1910 Fieldner and Davis[5] reported that large differences in volatile matter were secured in two U. S. Geological Survey laboratories, one of which was using natural gas and the other artificial gas. These differences were most noticeable in semibituminous coals, amounting in some cases to 3 and 4 per cent. Investigation showed that the large differences were obtained during the winter months when the pressure of natural gas was unusually low. This low pressure resulted in low flame temperatures and consequently in low volatile-matter determinations at the Pittsburgh laboratory, even though the height of flame was as specified in the official method. A later paper by Fieldner and Hall[6] showed the influence of temperature on volatile-matter determinations, and called attention to the fact that flame temperatures obtained with ordinary Bunsen burners using natural gas were lower than when manufactured gas was used, especially when the pressure fell off in the burners and the admixture of primary air was inadequate.

Joint Committee of A. C. S. and A. S. T. M.

These discoveries led the American Chemical Society and the American Society for Testing Materials to appoint a joint committee to investigate methods of sampling and analysis of coal and to submit a new report which should represent the best practice for commercial coal analysis. This joint committee, organized in 1911, consisted of W. A. Noyes, Chairman; Perry Barker, H. C. Dickinson, A. C. Fieldner, Frank Haas, W. F. Hillebrand, S. W. Parr, S. S. Voorhees and A. H. White. They held their first meeting in 1911 and appointed the following subcommittees:

Preparation of Samples, including loss of moisture in sampling.—A. C. Fieldner, chairman; Haas, Hillebrand, Voorhees, Parr and Barker.

Moisture.—W. F. Hillebrand, chairman; Fieldner, Parr.

Deterioration.—S. W. Parr, chairman; Fieldner, Haas, Dickinson.

Volatile Matter.—S. W. Parr, chairman; Fieldner, Haas, Dickinson.

Fixed Carbon and Ash.—S. W. Parr, chairman; Fieldner, Hillebrand.

Sulfur.—Perry Barker, chairman; Voorhees, Dickinson.

Phosphorus.—W.F. Hillebrand.

Ultimate Analysis.—A. C. Fieldner, chairman; Parr, White.

Calorimetric Determination.—H. C. Dickinson, chairman; Haas, Barker.

Interpretation and Computation.—Entire committee.

---

[5] A. C. Fieldner and J. D. Davis: Some Variations in the Official Method for the Determination of Volatile Matter in Coal. *Jnl. Ind. & Eng. Chem.* (1910) **2**, 304.

[6] A. C. Fieldner and A. E. Hall: Influence of Temperature on Determination of Volatile Matter in Coal. Eighth International Congress of Applied Chemistry (1912) **10**, 139.

These subcommittees at once began active work and in 1913 the joint committee presented a preliminary report.[7] With reference to the empirical nature of methods of coal analysis, Chairman Noyes made the following statement:

> It cannot be emphasized too strongly that in coal we have an extremely complex mixture of organic compounds, many of which are easily changed by oxidation and otherwise by exposure to the air, and containing water in such condition that a part of it is usually lost with the greatest ease, while the material after complete drying is excessively hygroscopic, probably even more so than calcium chloride. These actual conditions give rise to variations in analysis very far beyond those which most analysts, who have not studied the matter carefully, realize.
>
> In sampling and analyzing it is necessary to emphasize again the fact that coal is an extremely changeable substance, that all operations should be carried out as rapidly as possible, and that the analysis should be made as soon as practicable after the sample reaches the laboratory.
>
> *Moisture.*—It is probably impossible to secure an actually exact determination of the amount of moisture present in coals, especially in those of the subbituminous or lignite type, partly because some compounds in the coal decompose readily with loss of water, and partly because of the rather rapid oxidation of such coals when heated in the air. The determination of moisture is, therefore, to be considered as empirical rather than theoretical, and we should strive for conditions that should give concordant results, the agreement of different analysts being of more importance than a determination of the actual amount of moisture in the coal, since the calorific value of the "pure coal" depends on the method of determining moisture in such a manner that it is unimportant whether some of the moisture is counted as moisture or as a part of the coal.
>
> *Volatile Matter.*—The determination of volatile matter is empirical even to a much greater extent than that of moisture, and here again the point that must be aimed at is to secure conditions which can be accurately reproduced by different analysts. The most important change which seems necessary from the procedure recommended by the committee of the American Chemical Society in 1899 is the application of a gentle heat at first to those lignites which undergo a large mechanical loss if the full heat is applied at once.

This preliminary report was somewhat further revised and presented as a final report to both societies in June, 1915. It was published as Proposed Tentative Methods for the Sampling and Analysis of Coal in the Year Book of the American Society for Testing Materials for 1915 (Pages 596–624). The methods were finally adopted as standard methods of the American Chemical Society and the American Society for Testing Materials in 1916. They are practically the same as the methods of the U. S. Bureau of Mines, described in *Technical Paper* 8, revised as of 1913. Since 1916 three committees of the American Society for Testing Materials (Committee on Sampling of Coal, Committee on Sampling and Analysis of Coke, and Committee on Laboratory Sampling

---

[7] W. A. Noyes: Preliminary Report for Committee on Coal Analysis of American Society for Testing Materials and American Chemical Society. *Jnl. Ind. & Eng. Chem.* (1913) **5**, 517.

and Analysis of Coal) have been combined into one committee, D-5, on Coal and Coke, which is a permanent standing committee. From year to year additions and minor revisions of the standard methods are made. None of these changes have been radical, but all of them have been in the direction of more definitely standardizing conditions so that greater uniformity may be obtained in different laboratories.

In Canada the methods of the Fuel Testing Station of the Department of Mines are most generally used, at least with respect to the published analyses which have largely emanated from that laboratory and the laboratory of the Department of Scientific and Industrial Research of Alberta, at Edmonton, Alberta. These methods are essentially the same as those of the American Society for Testing Materials.

Until recent years none of the European countries have made any particular efforts towards standardization of methods of sampling and analyzing coal. During the last two years the Fuel Research Board in England has taken the initiative in standardizing British methods. There has been informal cooperation between the United States and England, and it is believed that eventually the English methods will be practically the same as ours.

Influence of Variations in Method on Published Analytical Results

Most of the published coal analyses of the United States have been made by the U. S. Bureau of Mines and its predecessors, the Technologic Branch and Fuel Testing Plant of the U. S. Geological Survey. Likewise, most of the published Canadian analyses have been made by the Dominion Department of Mines, which followed essentially the same analytical methods as the governmental laboratories of the United States. Analyses of mine samples are best adapted for classifying coal. The government samples were collected with special precautions to prevent change of moisture content, by shipping the samples in air-tight containers to the laboratory; also, the Government Fuel Testing Plant, in beginning its work in 1904, worked out a method of weighing the samples immediately upon removal from the containers, then air-drying them to bring the coal into approximate equilibrium as regards moisture with the air. Further precautions were taken to minimize moisture changes during sample preparation in the laboratory. Therefore the samples collected and analyzed by the U. S. Government may be taken as representative of the moisture content of the coal in the mine, provided the samples are taken from a newly exposed face so that the coal is not oxidized or weathered. Outcrop samples are usually weathered and oxidized. In general, the various state geological surveys also followed the procedure of the government laboratory in the years following 1904.

Analyses published prior to 1904 are unlikely to represent the true moisture content of coal as it occurs in the mine; and even after this date, analyses from other sources than the national government or state surveys are not likely to be representative with respect to moisture. Very often such samples were shipped to the laboratories in bags, boxes or other containers which did not preserve the moisture in the coal.

The other important modification of the American Chemical Society methods of 1899 was the standardization of the volatile-matter determination at 950° C. in 1913. The influence on analytical results of these major as well as minor changes in method will now be discussed with reference to the published analyses of the U. S. Geological Survey and the U. S. Bureau of Mines, since these analyses comprise the principal source of data for the classification of coals of the United States.

Air-drying Loss

The air-drying loss is not regarded as a significant or duplicatable determination in the work of the U. S. Geological Survey and U. S. Bureau of Mines. The main purpose of the determination has always been to bring the sample into approximate equilibrium with the atmosphere and to keep account of the moisture lost under the atmospheric conditions prevailing at the time of air-drying. With this method of air-drying in use there should be no appreciable change in moisture content while preparing and analyzing the small laboratory sample. The published figures for air-drying loss are of little value in coal classification because atmospheric conditions during air-drying vary widely. In the early days of the St. Louis fuel-testing work in 1904, the samples were air-dried to constant weight by simple exposure to the laboratory air. Later, the time of drying was shortened by using a special air-drying oven in which the laboratory air, preheated to 30° to 35° C., was circulated over the coal.

Examination of analyses of subbituminous coals and lignites made by the Bureau of Mines on samples from the same mine in different years shows widely different results in air-drying loss, although the total moisture as-received agrees fairly well. These differences are due to the fact that the humidity of the air, the size of the coal, and especially the period of drying affect the air-drying loss. Actually the samples are seldom dried to full equilibrium with the air, and the deviation from equilibrium varies in samples analyzed at different times. In recent years the Canadian Department of Mines and the Scientific and Industrial Research Council of Alberta have air-dried at room temperature and 60 per cent. humidity. Their results should be fairly comparable and of some significance in the classification of coal.

Moisture at 105° C.

The only change in the method for determining moisture since 1904 has been to increase the rate of circulating dry air through the oven from 8 to 10 times the oven volume in 1 hr., to 2 to 4 times the oven volume in 1 min. This change was made in 1912. The slower rate was found to be insufficient for maximum drying effect when analyzing high-moisture coals and lignite. The moistures reported in analyses of subbituminous coals and lignites bearing laboratory numbers below 14,000 are probably from 0.5 to 2.0 per cent. low, as compared to analyses represented by serial numbers above 14,000; *i. e.*, 20 per cent. moisture might have been reported in one case and 21 per cent. in the other. These differences are not important, since they apply only to the high-moisture coals. For practical purposes the total moisture results may be taken as comparable from 1904 to date.

Ash

The ash results also may be taken as comparable for the entire set of government analyses, although in the early years of the work thermocouples were not kept in the muffles to insure a uniform temperature.

Volatile Matter and Fixed Carbon

As explained earlier in this report, the method for determining volatile matter was materially changed in the 1913 revision of the American Chemical Society. The new revision was substantially the same as that introduced by the U. S. Bureau of Mines a few years earlier, as a result of finding large discrepancies between results of the Pittsburgh and Washington laboratories. Unfortunately, a considerable number of analyses of West Virginia semibituminous coals were made before it was discovered that the natural-gas burners at the Pittsburgh laboratory gave low results on volatile matter and correspondingly high results for fixed carbon. The following statement from pages 28 and 29 of U. S. Bureau of Mines *Bulletin* 22 explains these differences and gives corrections to be applied to make the results comparable:

> Volatile-matter determinations made in different laboratories may not agree closely, even though each laboratory conforms to the method recommended by the American Chemical Society. This method prescribes the size of the flame, but does not consider the variations in flame temperature resulting from differences in the composition of the gas used and in the pressure at which it is supplied to the burner. Hence the volatile-matter, and consequently the fixed-carbon determinations published in this bulletin are not directly comparable throughout, because the work was done in three different laboratories, under four different sets of conditions. In making comparsions the determinations should be considered in four groups, as follows:
>
> Group 1, laboratory Nos. 1 to 5145, inclusive. These determinations were made in the St. Louis laboratory, where gasoline gas was used for fuel.

Group 2, laboratory Nos. 5147 to 7100, inclusive. These determinations were made while the laboratory was in the Carnegie Technical Schools, Pittsburgh, Pa., where natural gas was used as fuel. There is no record of the pressure at which the natural gas was supplied to the burners, but this pressure was probably about 10 inches of water.

Group 3, laboratory Nos. 7101 to 9120, inclusive. These determinations were made after the removal of the laboratory to its present site, Fortieth and Butler Streets, Pittsburgh, Pa., where natural gas has been used for fuel. During the period of the determinations in this group, the low pressure of the gas at the burners gave much trouble. The pressure fluctuated between 1½ and 5 inches of water, apparently varying with the demands of certain industrial establishments that were taking gas from the same main.

Group 4, laboratory Nos. 9121 and over, were analyzed under the same conditions as group 3, except that the pressure of the gas at the burners was kept at 10 to 14 inches of water. With the use of the Tyrell burner and a polished platinum crucible a temperature of about 880° C. was maintained in the interior of the coke, at a point about 2 millimeters from the bottom of the crucible.

Comparisons of the analyses of samples of coal from the same mine show that the volatile matter and the fixed carbon determinations of group 1 and group 4 agree fairly closely; the variations are both plus and minus and as a rule within 1 per cent. The determinations of group 3, however, are distinctly lower in volatile matter and higher in fixed carbon than are those of group 1 and of group 4. The differences are about 3 per cent. for low-volatile semibituminous coals and anthracite, and decrease gradually, as the volatile matter in the coal increases, to about 1 per cent. for bituminous coals. The volatile matter determinations made while the laboratory was in the Carnegie Technical Schools (group 2) fall about midway between the determinations at the St. Louis laboratory (group 1) and those made with natural gas under low pressure (group 3).

The volatile matter of some lignite and subbituminous coal samples, designated in the table of analysis by an asterisk (*), was determined by the modified official method. These samples were given a preliminary heating of 4 minutes over a small flame, and a final heating of 7 minutes over a flame 20 centimeters high.

Sulfur

All sulphur determinations were made by the Eschka method and are comparable in the Government analyses.

Ultimate Analyses

All ultimate analyses were made by essentially the same method throughout the period and should be comparable.

Calorific Value

All calorific-value determinations were made with the oxygen bomb calorimeter. Since the beginning of the work in 1904, the technique has been improved, and the work since 1913 has been more accurate because of better standardization, the use of thermostatically controlled calorimeters and other minor improvements. It is believed that the

total change in absolute values due to change of fundamental standards did not exceed 0.6 per cent.

Normal Variations in Analytical Results by Present Standard American Method

The normal tolerances or permissible differences between two or more determinations made in different laboratories are given by the American Society for Testing Materials (A. S. T. M. Designation: D, 271–29), as shown in Table 1.

Table 1.—*Normal Tolerances between Determinations*

A. On Laboratory Samples, Crushed to Pass through an 840-Micron (No. 20) Sieve.	
Moisture:	
Under 5 per cent	0.3
Over 5 per cent	0.5
B. On Laboratory Samples, Crushed to Pass through a 250-Micron (No. 60) Sieve.	
1. Moisture:	
Under 5 per cent	0.3
Over 5 per cent	0.5
2. Ash:	
No carbonates present	0.3
Carbonates present	0.5
[Coals with more than 12 per cent. of ash, containing carbonate and pyrite	1.0
3. Volatile Matter:	
Bituminous coals	1.0
Lignites	2.0
Cokes (high-temperature)	0.4
4. Sulfur:	
Coal, Under 2 per cent	0.10
Coal, Over 2 per cent	0.20
Coke	0.05
5. Ultimate Analysis (Author's estimate):	
Carbon	0.4
Hydrogen	0.1
Nitrogen	0.1
6. Calorimetric Determination:	
Permissible differences	0.5%
7. Fusibility of Coal Ash:	
Permissible differences	50° C.

These permissible differences apply when the A. S. T. M. method is followed in all particulars. If the analyst deviates from that method, the differences may be much larger. Also, the differences apply only when the various laboratories work on different portions of the same finely pulverized sample. Errors due to taking, reducing and preparing the laboratory sample are frequently larger because of moisture changes

and nonuniform distribution of ash-forming material. In good sampling practice the ash should be correct to within 1 per cent.

Oxygen is not included in the table, since it is not determined directly but is calculated "by difference." It accumulates all the errors of the other determinations of the ultimate analyses. The absolute error is impossible to estimate, but the relative error may easily be 0.5 to 1 per cent.

Recommendations

It is recommended that the standard methods of sampling and analyzing coal as adopted by the American Chemical Society and the American Society for Testing Materials be considered as standard for the classification of coal for the following reasons:

1. The methods have been studied and gradually improved over a period of years and represent the best efforts of the leading coal chemists of America.
2. These methods are now used in practically all American laboratories.
3. By far the largest amount of available analytical data resulting from coal surveys, notably those of the U. S. Geological Survey and the U. S. Bureau of Mines, were obtained with the use of these methods.
4. In using analyses of mine samples for the classification of coal on the basis of composition as the coal occurs in the bed, care must be observed that these samples do not come from outcrops or near enough to the surface to be weathered and oxidized. Samples cut from the surfaces of old pillars are also likely to show deterioration. Samples for classification are best taken from the freshly exposed face of the coal bed.

Present Status of Ash Corrections in Coal Analysis*

By A. C. Fieldner† and W. A. Selvig,‡ Washington, D. C.

(New York Meeting, February, 1930)

For purposes of coal classification it is desirable to know the composition and calorific value of the pure coal substance; that is, of the coal free from its ash-forming minerals. Two methods suggest themselves: (1) To free the coal from most of the ash-forming material previous to the analysis; and (2) to analyze the coal as received and calculate the analysis, with certain more or less arbitrary corrections, to the pure-coal basis. Calculation methods are obviously more apt to give correct results with low-ash than high-ash coals. With coals of unusually high ash it would appear preferable to remove, previous to the analysis, as much of the inorganic matter as possible, by float-and-sink or acid-treating methods.

Effect of Ignition on Inorganic Constituents of Coal

The need of corrections in calculating analysis to the pure-coal basis is due to the fact that the ash, as determined by ignition, is not the same, in either weight or composition, as the inorganic mineral matter in the coal. On ignition a number of changes occur in the coal such as loss of water of hydration of the shaly material, conversion of pyrite (FeS_2) to ferric oxide (Fe_2O_3) with evolution of the sulfur as sulfur dioxide or possible fixation of more or less of the sulfur in the ash as calcium sulfate ($CaSO_4$), and decomposition of calcite ($CaCO_3$) to calcium oxide (CaO). The amount of sulfur from pyrite which remains in the ash as calcium sulfate depends on the amount of calcite and pyrite in the coal. Certain coals high in both calcite and pyrite show relatively large amounts of sulfate sulfur in the coal ash. The determination of total carbon and hydrogen in coal includes, besides the organic carbon and hydrogen, the carbon of inorganic carbonates such as calcite, and the hydrogen of the water of composition of some of the ash-forming minerals. For detailed discussion of corrections proposed for calculations of coal analyses to the "pure"

---

* Published by permission of the Director, U. S. Bureau of Mines.

† Chief Chemist, U. S. Bureau of Mines, and Chairman of Sectional Committee on Classification of Coal of American Standards Association.

‡ Associate Chemist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

or "unit coal" basis, reference should be made to publications of Parr and Wheeler[1] and of Tideswell and Wheeler.[2]

Formulas for Correcting Ash

Parr[3] defines unit coal as the pure coal substance considered altogether apart from extraneous or adventitious material which by accident or through natural causes may have become associated with the combustible organic substance of the coal. He proposes the following formula for estimating the non-coal substance:

$$\text{Non-coal} = M + A + \tfrac{5}{8}S + 0.08(A - \tfrac{10}{8}S) \qquad [1]$$

in which M is moisture,

A is ash as weighed, and

S is sulfur.

$\frac{5}{8}S$ restores the Fe_2O_3 as weighed in the ash to FeS_2 as weighed in the coal, 3 oxygens or 48 in the ash having been originally 4 sulfurs or 128 in the coal.

$\frac{10}{8}S$ represents the equivalent of Fe_2O_3 as weighed in the ash; that is, the Fe_2O_3 molecule, 160, is $\frac{10}{8}$ of the sulfur present in the coal.

$(A - \frac{10}{8}S)$ is the ash as weighed minus the Fe_2O_3.

0.08 is a constant applied to the iron-free ash to restore the water of hydration to the earthy matter less iron pyrites, thus representing the true amount of shaly constituent as weighed in the original coal.

To simplify calculation as well as to promote accuracy by compensating for the sulfur not in the pyritic form, Parr changes the fraction $\frac{21}{40}S$ to $\frac{22}{40}S$, or $0.55S$. Unit coal and unit-coal British thermal units are expressed by the following formulas:

$$\text{Unit coal} = 1.00 - (M + 1.08A + \tfrac{22}{40}S). \qquad [2]$$

$$\text{Unit-coal B.t.u.} = \frac{\text{indicated B.t.u.} - 5000S}{1.00 - (M + 1.08A + \tfrac{22}{40}S)}. \qquad [3]$$

The expression $5000S$ represents the heat derived from burning the sulfur to SO_2 and the iron to Fe_2O_3, and is 247 less than the actual heat of combustion of pyrite to partly compensate for the fact that some of the sulfur is in the organic form.

Tideswell and Wheeler assign a value of 10 per cent. for combined-water content of the mineral matter, as compared to Parr's 8 per cent.

---

[1] S. W. Parr and W. F. Wheeler: Unit Coal and the Composition of Coal Ash. Univ. Ill. Eng. Exp. Sta. *Bull.* 37 (1909).

[2] F. V. Tideswell and R. V. Wheeler: Pure Coal as a Basis for Classification. *Trans.* A. I. M. E. (1928) **76**, 200.

[3] S. W. Parr: The Classification of Coal. Univ. Ill. Eng. Exp. Sta. *Bull.* 180 (1928).

They recommend the following formula for calculating the total inorganic material from a dry-coal analysis:

$$\text{Total inorganic material} = 1.1(A - 1.25S_{pyr}) + 1.875S_{pyr}$$
$$= 1.1A + 0.5S_{pyr} \qquad [4]$$

The pure coal then becomes $100 - (1.1A + 0.5S_{pyr})$ [5]

Tideswell and Wheeler list a number of analyses giving sulfur forms of different coals and suggest that as the organic sulfur values range around 1 per cent., one might well assume that the pyritic sulfur amounts to the total sulfur less 1 per cent., or $S_{pyr.} = S_{total} - 1$.

They studied the effects of adding definite amounts of shale, pyrites, and calcium carbonate to coals of very low ash content and calculating the analyses to the usual ash-free basis and to the mineral-matter-free basis as expressed by their formula. It was found that elimination of carbon dioxide from calcite occurred only partially in the determination of ash and in the determination of total carbon. Another difficulty presented with coals having considerable calcite and pyrite was the fixation of varying amounts of sulfur in the ash as calcium sulfate. They therefore recommend the removal of calcium carbonate, previous to analysis, by a float-and-sink separation. Most coals, however, are practically free from carbonate and need not be treated. Tideswell and Wheeler conclude that when analyses of coal are required for purposes of classification, on the basis of ultimate analysis, the coals should be given a preliminary treatment before analysis, so as to reduce the ash to a reasonably low value (not exceeding, if possible, 4 to 5 per cent.). They suggest doing this by careful selection of the sample or by float-and-sink separation. This preliminary treatment gets rid not only of carbonate but also of much of the shaly and the coarser pyritic ingredients. As methods of correction are only approximately correct, it is obvious that a reduction of the ash-forming minerals previous to the analysis will lessen errors involved in calculation.

Parr[4] has proposed special corrections for coals containing a good deal of carbonate, as for instance over 0.3 per cent. CO_2. The ash determination is made at a temperature higher than usual so as to drive off all the CO_2. This is accomplished by blasting in a platinum crucible to constant weight and correcting the ash thus determined by adding the weight of CO_2 as obtained by an acid evolution method. As this high ignition temperature drives off any chlorine present, the amount of chlorine in the coal is also determined and added to the weight of ash. Parr's formula for coal containing carbonates and chlorine is as follows:

$$\text{Unit coal (calculated from dry-coal analysis)} = 1.00 - [(\text{Ash at high temp.} + CO_2 + Cl)1.08 + \tfrac{22}{40}S] \qquad [6]$$

---

[4] S. W. Parr: Univ. of Ill. *Bull.* 37 (1909) 25.

In a subsequent paper, Parr[5] proposed a modified method for the determination of ash of coals containing considerable amounts of carbonates and pyrite. On ignition of such coals, a good deal of the sulfur of the pyrite is retained in the ash by fixation as $CaSO_4$ by the calcite of the coal. The sample is ashed in the usual way at a temperature of 700° to 750° C. The ash, after cooling, is then moistened with a few drops of sulfuric acid (1:1) and after drying is heated at a temperature of 750° C. for 3 to 5 min., cooled, and weighed. All the calcium originally present in the coal as carbonate and partly changed to calcium oxide and calcium sulfate during the first ignition is now present as calcium sulfate. The weight of this ash is corrected, for the calcium sulfate thus formed, by subtracting from the ash as weighed three times the weight of carbon present as carbonate in the unburned coal, in order to restore the weight of the calcium sulfate formed to its equivalent of calcium carbonate. Further corrections may be applied to account for the loss of weight due to the oxidation of pyrite to ferric oxide and for loss of water of hydration of the shaly material. Fieldner[6] suggested combining these corrections in the following formula:

$$\text{Corrected ash} = \text{ash}_w - 3C_1 + \frac{5S}{8} + 0.08\left[\text{ash}_w - \left(\frac{34C_1}{3} + \frac{10S}{8}\right)\right] \qquad [7]$$

which simplifies to

$$\text{Corrected ash} = 1.08\ \text{ash}_w - \frac{293}{75}C_1 + \frac{21S}{40} \qquad [8]$$

(Note: If one-half of sulfur is assumed to be pyrite, the formula becomes

$$\text{Corrected ash} = 1.08\ \text{ash}_w - \frac{293}{75}C_1 + \frac{3}{4}S. \qquad [9]$$

where ash_w = percentage of ash as weighed after ignition at 750° C. with the addition of sulfuric acid

C_1 = percentage of carbon occurring as carbonate in the unburned coal

S = percentage of pyritic sulfur in the unburned coal

Parr[7] has shown also that it is possible to so ash a coal containing both pyrite and calcite as to drive off the sulfur from the pyrite at a temperature low enough not to fix the sulphur in the ash as calcium sulphate. This is done by first ashing at a temperature between 500 and 600° C. At this low temperature the carbon and pyritic sulphur are burned off leaving, however, the $CaCO_3$ practically undis-

[5] S. W. Parr: Chemical Study of Illinois Coals, Univ. of Ill. Coal Mining Investigations, *Bull.* 3 (1916).

[6] A. C. Fieldner: Notes on the Sampling and Analysis of Coal. *Tech. Paper* 76, U. S. Bureau of Mines (1914).

[7] S. W. Parr: Univ. of Ill. *Bull.* 37 (1909) 25.

turbed. After this preliminary ashing for 3 to 4 hr., the ash is finally placed in a hot muffle and ashed at a temperature of 800° to 850° C. to decompose the $CaCO_3$. According to Parr, this method gave good results but required considerable time and care.

Verifying Formulas with Float-and-sink Separations

One method of verifying formulas for ash corrections is to take a high-ash coal and subject it to a float-and-sink separation so that the float coal will be very low in ash and the sink coal very high in ash. If it can be assumed that the pure coal substance of the float and of the sink portions are identical, one could verify the accuracy of the empirical formula by comparisons of the calorific values of the pure coal of the two portions. It may be questionable, however, to assume that the pure coal of the float portion and that of the sink portion are identical. Parr justifies his formula for calculating pure coal by this method. He compares results obtained with 11 different coals[8] and finds excellent agreement between the calculated British thermal units of the pure coal of the float and of the sink portions, the differences ranging from 1 to 66 B.t.u.

The writers had occasion to treat a number of coals by the float-and-sink method and to analyze the float-and-sink portions. A comparison of the calorific value of the pure coal of the float and of the sink portions was made by the usual moisture-and-ash-free calculation and also by Parr's formula. The results are shown in Table 1. In general Parr's formula gave much more concordant results than the results as calculated on the usual moisture-and-ash-free basis, although not as concordant as those published by him; the differences ranged from 40 to 500 B.t.u. and averaged 175 B.t.u. by the Parr formula, as compared to a range of 100 to 1050 B.t.u. and an average of 579 B.t.u. by moisture-and-ash-free computation.

In using the formula $\frac{\text{B.t.u.}}{100 - \text{ash}}$, Table 1 shows that the calculated B.t.u. of the pure coal of the sink portions is in every case lower than that of the float portions. This is due to the fact that the weight of ash is lower than the true mineral content of the coal, making the denominator of the formula too large. In the case of Parr's formula, corrections are applied to the ash and the values for the sink portions more nearly agree with those for the float portions, though in some cases, as in the sink portions of coals G and S, the ash is apparently much over-corrected.

---

[8] S. W. Parr: The Classification of Coal. Univ. of Ill. Eng. Expt. Sta. *Bull.* 180 (1928).

TABLE 1.—*Calculated Heat Values of Pure Coal of Float and Sink Portions*

Coal Designation	State	County	Condition	Analyses, Dry-coal Basis: Ash, Per Cent.	Sulfur, Per Cent.	B.t.u.	Method of Calculating B.t.u. of Pure Coal: B.t.u. Moisture-and-ash-free		Unit-coal B.t.u. by Parr Formula	
A	Pennsylvania	Somerset	Float	2.9	0.77	15,280	15,740		15,800	
			Sink	26.0	1.69	11,070	14,960	−780	15,470	−330
B	Maryland	Allegany	Float	4.3	0.86	14,930	15,600		15,690	
			Sink	16.0	1.08	13,020	15,500	−100	15,790	+100
C	Pennsylvania	Clearfield	Float	2.9	0.87	15,330	15,790		15,860	
			Sink	29.2	5.25	10,540	14,890	−900	15,670	−190
D	Pennsylvania	Somerset	Float	3.5	0.62	15,220	15,770		15,840	
			Sink	21.5	4.53	11,960	15,240	−530	15,790	−50
E	Pennsylvania	Cambria	Float	2.5	1.02	15,360	15,750		15,820	
			Sink	25.0	4.59	11,370	15,160	−590	15,810	−10
F	Ohio	Meigs	Float	4.0	1.08	13,890	14,470		14,550	
			Sink	35.9	5.75	8,900	13,880	−590	14,830	+280
G	Illinois	Williamson	Float	3.7	2.41	14,040	14,580		14,700	
			Sink	34.6	7.38	9,270	14,170	−410	15,200	+500
H	Pennsylvania	Westmoreland	Float	4.0	1.0	14,740	15,350		15,440	
			Sink	29.3	3.3	10,410	14,720	−630	15,400	−40
HW	Pennsylvania	Westmoreland	Float	4.3	1.2	14,630	15,290		15,390	
			Sink	23.2	3.6	11,330	14,750	−540	15,280	−110
I	West Virginia	(New River coal)	Float	2.0	0.7	15,410	15,720		15,780	
			Sink	22.0	2.0	11,800	15,130	−590	15,570	−210
J	West Virginia	(Pocahontas coal)	Float	2.1	0.6	15,450	15,780		15,830	
			Sink	20.9	0.8	12,160	15,370	−410	15,740	−90
K	Ohio	Jefferson	Float	4.4	1.5	14,360	15,020		15,130	
			Sink	25.7	5.1	10,740	14,450	−570	15,100	−30
L.	West Virginia	Raleigh	Float	2.0	0.7	15,500	15,820		15,870	
			Sink	24.0	1.5	11,590	15,250	−570	15,720	−150
M	Pennsylvania	Mercer	Float	3.5	0.9	15,190	15,740		15,820	
			Sink	28.7	2.7	10,700	15,010	−730	15,650	−170
N	Illinois	Williamson	Float	4.5	1.8	13,930	14,590		14,700	
			Sink	26.8	3.3	10,460	14,290	−300	14,870	+170
O	Pennsylvania	Westmoreland	Float	4.0	1.1	14,740	15,350		15,450	
			Sink	35.8	4.2	9,280	14,450	−900	15,370	−80
P	Pennsylvania	Allegheny	Float	3.8	1.0	14,620	15,200		15,280	
			Sink	33.0	2.8	9,830	14,670	−530	15,430	+150
Q	Pennsylvania	Westmoreland	Float	3.6	0.8	14,750	15,300		15,380	
			Sink	24.5	1.3	11,290	14,950	−350	15,410	+30
R	Pennsylvania	Somerset	Float	3.3	0.5	15,270	15,790		15,850	
			Sink	27.4	6.1	10,700	14,740	−1050	15,500	−350
RW	Pennsylvania	Somerset	Float	3.5	0.5	15,240	15,790		15,860	
			Sink	24.0	5.0	11,400	15,000	−790	15,630	−230
S	Illinois	(Mixture from 7 mines)	Float	4.5	2.6	14,140	14,810		14,950	
			Sink	32.9	4.7	9,740	14,520	−290	15,360	+410
						Average	difference	579		175

A similar investigation of the divergence of moisture-and-ash-free B.t.u. and of unit-coal B.t.u. (by Parr formula) for 14 different coal samples before and after washing was made by the Fuel Testing Laboratories of the Canadian Department of Mines.[9] The results are given in Table 2. In both methods of calculation the majority of the washed coals indicate a higher "pure" or "unit" value than the unwashed coals. The average (neglecting the algebraic sign) difference by the moisture-and-ash-free method is 130 and by the Parr method 110 B.t.u.

TABLE 2.—*Comparison of Moisture-and-ash-free and Unit-coal Britis Thermal Units for Unwashed and Washed Coal*

Sample No.	Condition of Coal	Moisture-and-ash-free B.t.u.	Difference	Unit-coal B.t.u.	Difference
M11 M211	Unwashed Washed	15,050 15,260	+210	15,500 15,590	+ 90
M5 M205	Unwashed Washed	14,730 14,930	+200	14,900 15,060	+160
M6 M206	Unwashed Washed	14,660 14,800	+140	14,890 14,950	+ 60
M9 M209	Unwashed Washed	14,000 14,160	+160	14,430 14,510	+ 80
M7 M207	Unwashed Washed	14,030 14,180	+150	14,420 14,510	+ 90
M10 M210	Unwashed Washed	14,240 14,210	− 30	14,730 14,500	−230
M4 M204	Unwashed Washed	14,530 14,590	+ 60	14,830 14,630	−200
M1 M201	Unwashed Washed	14,750 14,880	+130	15,020 15,100	+ 80
M3 M203	Unwashed Washed	15,160 15,280	+120	15,470 15,490	+ 20
M14 M214	Unwashed Washed	13,580 13,690	+110	13,880 13,900	+ 20
M15 M215	Unwashed Washed	13,790 14,090	+300	14,250 14,470	+220
M36 M236	Unwashed Washed	14,740 14,720	− 20	14,890 14,800	− 90
M37 M237	Unwashed Washed	14,780 14,740	− 40	15,010 14,880	−130
M13 M213	Unwashed Washed	14,840 15,010	+170	15,030 15,130	+100
Average (neglecting algebraic sign)			130		110

[9] Private communication from J. H. H. Nicolls, chemist.

The difference by the two methods is small and would scarcely justify the more complicated method of correcting ash; however, it is likely that the differences in ash between the washed and unwashed coal are relatively small. It is certain that when the differences are large, as in comparing high-ash sink coal with very low-ash float (Table 1), the application of the Parr formula secures much more concordant results than the simple moisture-and-ash-free correction.

Corrections for Organic and Pyritic Sulfur

As previously mentioned, Tideswell and Wheeler propose the assumption that the pyritic sulfur amounts to the total sulfur less 1 per cent. In Table 3 are given sulfur forms as determined by the writers on a number of coals. The organic sulfur ranged from 0.43 to 1.85 per cent., the average for all the coals being 0.83 per cent. It is interesting to note that from 40 to 60 per cent. of the sulfur is present in the organic form in the majority of the coals. The average proportion of organic sulfur in the coals containing 1 per cent. or less of sulfur was 75 per cent., and in coals containing more than 1 per cent. sulfur was 45 per cent.

Table 3.—*Sulfur Forms of Various Coals, Percentage of Moisture-free Coal*

Coal Designation	State	County	Sulfur				Ratio Organic S, Total S, Per Cent.
			Pyritic, Per Cent.	Sulfate, Per Cent.	Organic, Per Cent.	Total, Per Cent.	
A	Pennsylvania	Somerset	0.19	0.02	0.57	0.78	73
B	Maryland	Allegany	0.18	0.01	0.67	0.86	78
C	Pennsylvania	Clearfield	1.12	0.03	0.75	1.90	39
D	Pennsylvania	Somerset	1.43	0.03	0.54	2.00	27
E	Pennsylvania	Cambria	0.56	0.01	0.65	1.22	53
F	Ohio	Meigs	1.61	0.04	0.86	2.51	34
G	Illinois	Williamson	2.17	0.04	1.80	4.01	45
H	Pennsylvania	Westmoreland	0.73	0.05	0.74	1.52	49
HW	Pennsylvania	Westmoreland	0.64	0.05	0.94	1.63	58
I	West Virginia	(New River coal)	0.39	0.03	0.57	0.99	58
J	West Virginia	(Pocahontas coal)	0.06	0.01	0.52	0.59	88
K	Ohio	Jefferson	1.20	0.07	1.01	2.28	44
L	West Virginia	Raleigh	0.22	0.01	0.58	0.81	72
M	Pennsylvania	Mercer	0.59	0.04	0.77	1.40	55
N	Illinois	Williamson	0.87	0.03	1.26	2.16	58
O	Pennsylvania	Westmoreland	0.90	0.01	0.89	1.80	49
P	Pennsylvania	Allegheny	0.61	0.00	0.80	1.41	57
Q	Pennsylvania	Westmoreland	0.11	0.01	0.70	0.82	85
R	Pennsylvania	Somerset	1.42	0.03	0.48	1.93	25
RW	Pennsylvania	Somerset	1.02	0.02	0.43	1.47	29
S	Illinois	(Mixture from 7 mines)	1.76	0.06	1.85	3.67	50

For all the coals in Table 3 the average proportion of organic sulfur was 0.54 per cent. The writers therefore believe that a correction

based on the assumption that one-half the total sulfur is pyritic will be nearer the truth than either Parr's assumption of nearly 100 per cent. pyritic sulfur or Tideswell and Wheeler's assumption of 1 per cent. organic sulfur in all coals.

On the basis of one-half pyritic sulfur and one-half organic sulfur, Parr's formula for unit coal would be modified as follows:

Parr's formula assuming that all sulfur is pyritic is:

$$\text{Non-coal} = M + A + \tfrac{5}{8}S + 0.08(A - \tfrac{10}{8}S) \qquad [10]$$

Modification for one-half sulfur to be pyritic:

$$\text{Non-coal} = M + A + \tfrac{5}{8}\cdot\tfrac{1}{2}S + \tfrac{1}{2}S + 0.08(A - \tfrac{10}{8}\cdot\tfrac{1}{2}S)$$

Simplifying:

$$\text{Non-coal} = M + 1.08\,A + \tfrac{61}{80}S$$

Sulfur can be rounded off to $\tfrac{3}{4}S$:

$$\text{Non-coal} = M + 1.08\,A + \tfrac{3}{4}S$$

and

$$\text{Unit-coal} = 1.00 - (M + 1.08\,A + \tfrac{3}{4}S) \qquad [11]$$

Likewise, the numerator of Parr's formula for unit-coal B.t.u. should be modified by subtracting from the indicated B.t.u. the sum of one-half the heat of combustion[10] of 1 g. of sulfur as pyrite, burning to SO_2 and Fe_2O_3 (5247 B.t.u.), and one-half the heat of combustion of 1 g. of sulfur in the free condition[11] burning to SO_2 (4050 B.t.u.), as in the following:

$$\text{Unit-coal B.t.u.} = \frac{\text{Indicated B.t.u.} - (\tfrac{5247}{2}S + \tfrac{4050}{2}S)}{1.00 - (M + 1.08A + \tfrac{3}{4}S)}$$

which simplifies to

$$\text{Unit-coal B.t.u.} = \frac{\text{Indicated B.t.u.} - 4650S}{1.00 - (M + 1.08A + \tfrac{3}{4}S)} \qquad [12]$$

Correction for Water of Constitution

Parr proposes a factor of 1.08 to be applied to the ash as determined to correct for loss of water of constitution of the ash-forming minerals. This factor also was intended to include slight additional losses during ashing. Tideswell and Wheeler analyzed a roof shale and found 91 per cent. ash and 9 per cent. combined water. Stansfield and Sutherland also recommend 1.1 as the factor for ash correction. They found this factor to vary from 1.05 to 1.20 for Alberta coals.[12] This value for

---

[10] E. E. Somermeier: Sulphur in Coal. *Jnl.* Amer. Chem. Soc., (1904) **26**, 566.

[11] S. W. Parr and W. F. Wheeler: Unit Coal and the Composition of Coal Ash. Univ. of Ill. *Bull.* 37 (1909) 11.

[12] E. Stansfield and J. W. Sutherland: The Classification of Canadian Coals. *Canadian Mining & Metallurgical Bull.* (1929) No. 210, 1158.

TABLE 4.—*Partial Analysis of Shales Associated with Various Coals*

State and County	Laboratory Number	Moisture at 105° C., Per Cent.	Ash, Per Cent.	CO_2, Per Cent.	Combined Water, Per Cent.	Factor to Apply to Ash Value for Combined Water	Factor to Apply to Ash Value for Combined Water Plus CO_2
Alabama							
Walker	36,624	1.0	86.7	2.5	3.9	1.045	1.074
Colorado							
Huerfano	36,706	2.0	90.8	0.9	4.3	1.047	1.057
Huerfano	36,707	2.0	92.7	1.7	2.7	1.029	1.047
Las Animas	36,783	3.9	87.3	0.1	6.4	1.073	1.074
Illinois							
Franklin	36,428	1.5	92.2	1.6	3.6	1.031	1.056
Jefferson	36,995	0.5	92.2	2.8	3.5	1.030	1.068
Macoupin	36,597	0.9	91.4	0.0	4.5	1.041	1.041
Macoupin	36,609	0.9	91.4	0.0	4.9	1.053	1.053
Madison	36,616	7.8	87.4	0.0	2.3	1.026	1.026
Madison	36,617	11.8	82.2	0.1	3.8	1.046	1.048
Madison	36,618	8.0	85.5	0.1	3.8	1.044	1.046
Sangamon	36,498	5.1	83.6	0.0	5.0	1.060	1.060
Williamson	36,501	3.3	87.7	0.0	4.4	1.050	1.050
Williamson	36,596	1.1	92.3	2.1	3.8	1.041	1.064
Williamson	36,608	1.1	92.3	2.1	3.7	1.041	1.063
Williamson	36,762	0.9	92.7	0.1	4.1	1.044	1.045
Williamson	36,886	0.8	92.8	2.1	3.5	1.038	1.060
Williamson	36,887	0.6	93.2	1.4	4.1	1.044	1.059
Williamson	37,157	1.1	92.6	1.0	3.6	1.039	1.049
Indiana							
Sullivan	36,561	3.3	89.1	2.2	2.7	1.030	1.055
Kentucky							
Union	36,857	0.9	88.4	2.0	3.7	1.042	1.065
New Mexico							
Santa Fe	36,583	1.7	90.5	0.3	7.2	1.080	1.083
Pennsylvania							
Allegheny	36,385	1.8	92.6	0.0	4.2	1.045	1.045
Allegheny	36,462	0.4	91.5	0.8	4.7	1.051	1.060
Allegheny	36,464	0.6	90.8	2.0	3.7	1.041	1.063
Allegheny	36,593	0.9	91.3	1.7	5.2	1.057	1.076
Allegheny	36,677[a]	1.0	86.9	3.6	4.7	1.045	1.096
Allegheny	36,678[a]	0.4	88.7	1.0	5.3	1.060	1.071
Allegheny	36,688	0.7	90.4	0.0	6.4	1.071	1.071
Allegheny	36,899[a]	1.3	89.5	0.1	6.1	1.068	1.069
Allegheny	36,900[a]	0.9	88.6	0.2	5.6	1.063	1.064
Allegheny	36,905	0.5	92.7	0.0	4.9	1.053	1.053
Allegheny	37,310	0.5	93.7	0.0	3.7	1.039	1.039
Allegheny	37,311	0.2	92.6	1.7	3.7	1.040	1.058
Armstrong	36,671[a]	0.9	89.8	1.7	4.9	1.054	1.073
Armstrong	36,672[a]	0.9	89.7	1.8	4.6	1.051	1.071
Armstrong	36,673[a]	0.8	91.3	1.3	4.3	1.047	1.061
Armstrong	36,674[a]	0.9	88.4	3.6	4.5	1.051	1.104
Berks	36,610	0.6	95.4	0.0	4.2	1.044	1.044
Cambria	36,675[a]	0.7	94.8	0.1	3.6	1.038	1.039
Cambria	37,006[a]	1.3	90.2	0.9	6.5	1.077	1.082
Cambria	37,007[a]	0.8	91.4	0.7	6.4	1.070	1.078
Cambria	37,008[a]	1.3	90.2	2.0	5.3	1.059	1.081
Cambria	37,009[a]	1.2	90.3	1.5	4.8	1.053	1.069
Cambria	37,043	0.7	89.3	0.1	4.0	1.045	1.046
Cambria	37,044	0.8	91.3	0.0	4.9	1.054	1.054
Cambria	37,045	0.5	90.0	2.5	3.9	1.043	1.071
Cambria	37,046	0.8	91.1	1.8	5.0	1.055	1.075
Cambria	37,047	0.8	91.8	0.0	5.1	1.056	1.056
Cambria	37,048	0.8	91.4	0.4	4.9	1.054	1.058
Clearfield	36,562	0.8	89.0	0.1	4.7	1.053	1.054

TABLE 4.—(*Continued*)

State and County	Laboratory Number	Moisture at 105° C., Per Cent.	Ash, Per Cent.	CO_2, Per Cent.	Combined Water, Per Cent.	Factor to Apply to Ash Value for Combined Water	Factor to Apply to Ash Value for Combined Water Plus CO_2
Fayette	36,467	2.1	84.4	0.2	6.4	1.076	1.078
Fayette	36,468	1.0	86.5	5.4	4.4	1.050	1.113
Fayette	36,916[a]	1.0	89.1	0.9	5.6	1.063	1.073
Fayette	36,917[a]	1.1	88.0	1.0	5.8	1.066	1.077
Fayette	36,919[a]	1.1	89.2	1.0	5.4	1.060	1.072
Fayette	36,920[a]	1.0	87.9	2.0	6.8	1.078	1.100
Fayette	36,921[a]	1.0	86.2	2.0	7.2	1.083	1.107
Fayette	36,922[a]	1.0	88.3	1.5	6.2	1.070	1.087
Fayette	36,923	1.1	87.2	1.3	5.7	1.065	1.080
Indiana	36,492	0.7	91.5	0.0	6.2	1.068	1.068
Indiana	36,493	0.5	93.5	0.0	5.3	1.057	1.057
Indiana	36,494	0.8	87.6	4.6	4.8	1.055	1.107
Indiana	37,001[a]	0.8	89.6	0.5	4.5	1.050	1.056
Indiana	37,002[a]	0.8	90.0	1.2	5.1	1.057	1.070
Indiana	37,003[a]	1.0	88.9	0.8	4.9	1.055	1.064
Jefferson	36,996[a]	1.2	89.0	1.5	5.2	1.058	1.075
Jefferson	36,997[a]	1.4	88.6	1.9	5.5	1.062	1.084
Somerset	37,051	1.3	91.8	1.4	4.8	1.052	1.068
Washington	36,660	1.3	87.7	0.0	7.4	1.084	1.084
Washington	36,682	1.0	91.7	0.1	7.0	1.076	1.077
Washington	36,683	1.0	90.0	0.1	5.1	1.056	1.058
Washington	36,684	1.0	90.5	0.1	6.1	1.067	1.068
Washington	36,685	1.0	89.6	0.1	6.0	1.067	1.068
Washington	36,897[a]	1.3	84.6	0.1	6.6	1.078	1.079
Washington	36,898[a]	1.2	85.1	0.2	5.8	1.068	1.070
Washington	36,918[a]	1.2	89.7	1.5	6.5	1.073	1.089
Washington	36,924[a]	1.3	88.9	1.7	5.0	1.056	1.075
Westmoreland	36,901[a]	2.5	84.1	0.2	7.2	1.086	1.088
Utah							
Carbon	36,382	5.3	82.3	4.4	2.9	1.036	1.089
Summit	36,375	12.6	70.3	4.6	3.8	1.052	1.115
Washington							
Kittitas	36,903	[illegible].4	[illegible].0	0.0	6.1	1.068	1.068
West Virginia							
Brooke	36,665	3.6	85.9	0.0	5.8	1.067	1.067
Brooke	36,666	2.8	83.0	0.0	6.4	1.077	1.077
Brooke	36,667	3.2	88.6	0.1	6.1	1.069	1.070
Logan	36,598	0.8	90.9	1.5	4.6	1.051	1.065
Logan	36,599	1.6	90.1	0.2	5.7	1.063	1.065
Logan	36,915	0.6	89.9	2.1	4.6	1.051	1.079
McDowell	36,526	0.4	92.7	2.6	2.8	1.030	1.058
McDowell	36,528	0.3	92.9	2.8	3.0	1.033	1.063
McDowell	36,532	0.2	94.2	0.9	4.3	1.046	1.055
McDowell	36,534	0.3	93.2	1.7	4.1	1.044	1.062
McDowell	36,535	0.2	93.4	1.2	4.0	1.043	1.056
Marion	36,497	0.8	84.9	3.3	4.9	1.058	1.097
Tucker	36,652	0.5	91.7	2.6	4.4	1.048	1.076
Tucker	36,523	1.1	91.0	1.4	4.3	1.047	1.063
Wyoming							
Lincoln	36,600	0.5	90.0	4.4	3.8	1.042	1.091
Lincoln	36,601	2.6	91.1	0.6	4.2	1.045	1.052
Lincoln	36,702	2.5	92.1	2.2	2.9	1.032	1.055
Lincoln	36,703	2.9	91.6	2.1	3.1	1.034	1.057
Lincoln	36,893	3.3	90.7	2.0	3.3	1.036	1.058
Sweetwater	36,912	1.4	90.1	3.5	3.4	1.038	1.077
Sweetwater	36,913	3.6	88.7	0.0	4.9	1.055	1.055
Sweetwater	36,914	1.1	89.1	5.7	3.6	1.040	1.104
Average					4.8	1.053	1.068

[a] Sample collected and analyzed by Alden H. Emery and R. DeChicchis.

combined water is higher than was obtained by Selvig,[13] who made partial analyses, including combined water, of a large number of shales associated with various coals of the United States; these analyses are shown in Table 4. It will be noted that the average combined water of the 104 shales listed in the table is 4.8 per cent. The last two columns give the factors for correcting the ash for combined water, and for combined water plus CO_2, respectively. The average factor of 1.053 is appreciably less than the 1.08 factor proposed by Parr and the 1.10 factor proposed by Tideswell and Wheeler. To correct for combined water plus carbon dioxide the table shows that the ash would be multiplied by 1.07; however, this would give erroneous results in case of coals having considerable calcite. With such coals it would appear best to determine the CO_2 directly and add the CO_2 content to the ash as determined after correcting for the combined water. These corrections, of course, are based on the assumption that the mineral ash-forming constituents of coal are practically the same as those found in shale associated with the coal. From Selvig's data, Parr's factor of 1.08 is preferable to the 1.10 suggested by Tideswell and Wheeler. However, 14 out of the 21 coals listed in Table 1 would show less deviation if 1.1 factor were used in place of 1.08, and 1.1 lends itself better to easy computation.

Correction for Ultimate Analysis

The presence of pyrite, carbonates, and water of constitution of clays and shale also seriously affects the ultimate analysis of impure coal for the purpose of classification. Tideswell and Wheeler[14] conducted some experiments in which known quantities of pyrite and of calcium carbonate were added to low-ash coal, the mixture was then subjected to ultimate analysis, and the results were computed to both ash-free and mineral-matter-free bases. They found that correction for pyritic sulfur gave distinctly better agreement than was obtained by disregarding pyrite as is done in the simple ash-free correction.

Correction for carbonate was more complicated and did not give good agreement, the reason being that even with the most careful combustion analysis the elimination of carbon dioxide was incomplete when tried on synthetic mixtures of coal and carbonate. For this reason they favor removal of as much of the mineral matter as possible by float-and-sink or other mechanical separation and then correcting for the residual mineral matter as follows:

---

[13] W. A. Selvig: Chemical Analysis of Rock-dusting Materials. A chapter of *Bull.* 21, *Min. & Met. Investigations*, Carnegie Institute of Technology.

[14] F. V. Tideswell and R. V. Wheeler: *Loc. cit.*, 207–210.

Pyrites $= 1.875S_{pyr}$ or 2.14 Fe or $1.875(S - 1)$
Pyritic ash $= 1.25S_{pyr}$ or 1.43 Fe or $1.25(S - 1)$
Shale ash $= A$ (total ash) $- 1.25S_{pyr}$
Shale $= \left(1 + \frac{w}{100}\right)(A - 1.25S_{pyr})$, where w = combined water in shale, per cent. on shale ash.

$$\text{Total inorganic material} = \left(1 + \frac{w}{100}\right)(A - 1.25S_{pyr}) + 1.875S_{pyr}$$

If the combined water associated with the shale ash is assumed to be 10 per cent., the total inorganic material $= 1.1(A - 1.25S_{pyr}) + 1.875 = 1.1A + 0.5S_{pyr}$. The pure coal is then given by $100 - (1.1A + 0.5S_{pyr})$.

The carbon values may be corrected directly to this basis, as also may be, if desired, the nitrogen and organic sulphur. Hydrogen needs a preliminary correction for water of hydration of shale (assumed to be 10 per cent. on shale ash).

$$\text{Water} = \frac{w}{100} - (A - 1.25S_{pyr}) = \frac{1}{10}(A - 1.25S_{pyr}).$$

$$H_0 = \frac{100(h - \frac{1}{90}\text{ shale ash})}{100 - (\text{shale} + \text{pyrites})} \text{ or } \frac{100[h - \frac{1}{90}(A - 1.25S_{pyr})]}{100 - (1.1A + 0.5S_{pyr})}$$

$$\left.\begin{array}{l} C_0 \\ N_0 \\ S_0\,(\text{organic}) \end{array}\right\} = \frac{100C \text{ (or } N \text{ or } S_{org})}{100 - (\text{shale} + \text{pyrites})} \text{ or } \frac{100C \text{ (or } N \text{ or } S_{org})}{100 - (1.1A + 0.5S_{pyr})}$$

$$O_0 = 100 - (C_0 + H_0 + N_0 + S_0).$$

In practice, analyses are now generally made on undried samples. The further correction for moisture content should be added to the formulas.

In the correction formula above it is assumed by Tideswell and Wheeler that the calcium carbonate remains undecomposed. They found that if they assumed all the calcium carbonate to be decomposed in analysis the errors were larger than when they assumed it to remain undecomposed. Evidently more research needs to be done on a study of the decomposition of calcium carbonate in coal mixtures, during ultimate analysis, before any degree of certainty is attached to these corrected values.

Conclusions

It is accepted generally that coal analyses and calorific values should be computed to a "pure-coal" or "unit-coal" basis for use in classifying coal for some purposes. Computation to a "moisture-and-ash-free" basis is the simplest procedure and is in most common use. The large number of published analyses of the United States and Canadian Govern-

ments are on this basis. This method is admittedly in error because of differences in weight of the inorganic mineral matter in the coal and the resulting ash, but for ordinary variations in ash, say from 5 to 12 per cent., the resulting divergences probably are not of sufficient magnitude to materially affect the classification of a coal. Nevertheless, as Parr and others have shown, corrections for pyritic sulfur and for water of constitution of shaly matter do give better agreement in unit-coal values than the pure-coal values based on uncorrected ash, especially in comparing the high-ash and low-ash fractions of the same sample of coal. The authors, therefore, have collected further experimental data on the subject from which the following conclusions are drawn:

1. Comparison of moisture-and-ash-free and unit-coal B.t.u. of the float-coal and sink-coal fractions, from float-and-sink tests of 21 different coals, showed an average difference of 579 B.t.u. by the moisture-and-ash-free formula and 175 B.t.u. by the Parr formula. The float-coal ash ranged from 2.0 to 4.5 per cent., and the sink-coal ash ranged from 16 to 35.9 per cent. (U. S. Bureau of Mines tests).

2. Comparison of moisture-and-ash-free and unit-coal B.t.u. of washed and unwashed portions from 14 different coals showed an average difference of 130 B.t.u. by the moisture-and-ash-free formula and 110 B.t.u. by the Parr formula. In these samples the differences of ash were in the usual commercial range (Canadian Bureau of Mines tests).

3. Examination of 21 different coals ranging in total sulfur from 0.59 to 4.01 per cent. showed that from 30 to 70 per cent. of the total sulfur was present as organic sulfur, the average per cent. being 54.

4. The Parr formula may be improved by assuming only one-half the sulfur to be present as pyrite, instead of all the sulfur.

5. Analysis of 104 samples of shales associated with coal beds in various parts of the United States showed combined water varying from 2.7 to 7.4 per cent. The average was 4.8. Carbon dioxide content of the same shales averaged 1.02 per cent. The average factor to apply to the ash of these shales to correct them for water of hydration and the carbon dioxide is 1.07. This is less than Parr's 1.08; however, other volatile constituents such as chlorides will probably raise the factor to 1.08, and further investigation may show 1.10 to be preferable.

6. It is suggested that Parr's formula be modified to read as follows:

$$\text{Unit-coal B.t.u.} = \frac{\text{Indicated B.t.u.} - 4650S}{1.00 - (M + 1.08A + \tfrac{3}{4}S)}$$

This modification assumes one-half the sulfur to be present as pyrite and one-half as organic sulfur.

7. It is a question whether any classification can be devised which will draw such fine distinctions as to require a corrected ash value. It must be kept in mind that although the corrected ash values by the Parr

formula give more concordant values when averaging a group of samples, yet the identical samples often deviate considerably, due to the variation of water of composition of inorganic constituents from 4 to 14 per cent., to varying amounts of carbonate and chlorides, and to fixing more or less sulfur by the interaction of pyrite and calcium carbonate. The average difference of unit-coal B.t.u. by the Parr formula in Table 1 is 175, but five samples in 21 vary more than 200 B.t.u. between the float coal and sink coal.

DISCUSSION

G. H. Cady, Urbana, Ill. (written discussion*).—It is of interest to compare the results obtained by the Parr formula with those obtained by using the modification of the Parr formula as proposed by Fieldner and Selvig in paragraph 6 of their conclusions when applied to Illinois coals.

The Illinois State Geological Survey has for many years presented the unit coal B.t.u. values in connection with all publication of analyses. A sufficient number of analyses have been made of the most important coal in many of the counties and all the counties in which mines are numerous, so that a rough standard unit coal value has been obtained for the coal in many local areas. Variations from this standard value of more than 100 units are regarded askance, and so far as our own laboratory is concerned, that is the Testing Laboratory of the University of Illinois, conducted by J. M. Lindgren, departures of more than 100 B.t.u. from the standard value require substantiation by repetition of the B.t.u. determinations before they are accepted. In fact, the average variation for many hundreds of analyses runs about 63 per cent.

Since the proposal of the modification of the Parr formula by the authors all analyses of Illinois coals in our files, totaling more than1000, have been recalculated in accordance with the modified formula and a comparative study has been made of the unit coal values as determined by each formula. The results will be briefly summarized.

Twenty-three county average unit coal values were studied (Table 5), each being based on an average of at least two analyses of samples collected in at least each of two mines. In some instances as many as 18 mines are represented. Examination of the variations from the average, irrespective of sign, shows that for the 23 counties the average variation of the individual mine averages from the county average by the Parr formula is 63.07 for the authors' formula 68.9. The smallest variation by the Parr formula is 16.8 units and for the authors' formula 7.8, the same county being smallest in both cases. The largest variation for the Parr formula is 117.4 units, the authors' formula giving an average departure of 141.8 units for the same county. Out of the 23 counties the Parr formula gives a smaller average variation than the Fieldner formula in 17 instances, the authors' formula being smaller in the remaining six. The range of variation above and below the county average for the Parr formula shows a maximum of 514 units; for the authors' 661 units, being the same county. The Parr formula gives five counties having a variation of mine averages of more than 300 units, whereas the authors' formula shows nine such counties. From the standpoint of county and mine averages it is quite apparent the advantage is with the Parr formula.

In the case of actual analyses and county averages the same general relationships obtain between the results gained by application of the two formulas. We have 18 analyses of No. 6 coal from Christian County. The variation from the county

* Published by permission of the Chief of the Illinois State Geological Survey.

TABLE 5.—*Twenty-three "Unit Coal" Values in Illinois*

By county and seam by Parr, and Fieldner and Selvig formulas; also maximum variation of mine average from county average range of variation and average variation irrespective of sign.

County	Seam	Unit Coal, Parr	Variation			Average Variation	Unit coal, Fieldner and Selvig	Variation			Average Variation
			Negative	Positive	Range			Negative	Positive	Range	
Christian	6	14,387	−87	+190	277	94.2	14,559	−96	+195	291	98.5
Clinton	6	14,329	−120	+185	305	76.0	14,476	−95	+231	326	93.5
Franklin	6	14,553	−148	+137	285	66.3	14,611	−167	+177	344	68.6
Fulton	5	14,495	−226	+288	514	177.4	14,642	−351	+310	661	141.8
Gallatin	6	15,133	−53	+76	129	62.4	15,297	−66	+129	195	68.2
Henry	1	14,507	−103	+55	158	53.0	14,724	−16	+79	95	37.7
Jackson	2	14,819	−114	+104	218	56.0	14,869	−119	+376	495	102.3
Jackson	6	14,608	−251	+5	256	85.3	14,699	−17	+281	298	105.3
Macoupin	6	14,256	−248	+152	400	93.7	14,430	−250	+155	405	103.1
Madison	6	14,356	−51	+65	116	34.7	14,553	−64	+67	131	40.1
Marion	6	14,589	−25	+41	66	26.6	14,754	−0	+18	18	9.6
Mercer	1	14,515	−166	+125	291	96.5	14,717	−177	+142	319	105.2
Montgomery	6	14,298	−169	+235	404	115.2	14,486	−185	+310	495	115.0
Peoria	5	14,615	−69	+77	146	37.2	14,758	−71	+75	146	41.0
Perry E.	6	14,370	−80	+115	195	59.0	14,479	−87	+57	144	36.2
Perry W.	6	14,335	−71	+116	187	66.6	14,469	−57	+84	141	46.3
Randolph	6	14,347	−119	+86	205	62.2	14,578	−201	+67	268	80.5
Sangamon	5	14,452	−41	+62	103	25.0	14,629	−122	+50	172	31.2
Sangamon	6	14,357	−70	+22	92	28.0	14,541	−39	+148	187	42.8
St. Clair	6	14,461	−156	+240	396	78.3	14,628	−245	+263	508	95.4
Vermilion	6	14,592	−58	+86	144	39.1	14,704	−76	+81	157	47.7
Williamson	6	14,619	−282	+111	393	61.3	14,707	−177	+172	349	62.1
Vermilion	7	14,738	−20	+9	29	16.8	14,865	+5	+13	18	7.8
		Average			230.8	63.07				267.9	68.69

average varies in the case of the Parr formula from 15 to 244 units, in the case of the Fieldner formula from 3 to 260 units. The average variation irrespective of sign is 92.4 units for the results obtained by the Parr formula, and 95.1 units for results obtained by the authors' formula.

Twenty-one analyses of No. 6 coal in Clinton County showed an average variation of 96.2 units using the Parr formula and 109.9 using the authors' formula.

Thirty-four analyses of No. 6 coal in Macoupin County showed an average variation from the county B.t.u. average by the Parr formula of 71.8 units and by the authors' formula of 80.7 units. Eleven mines are represented and of the mine averages only one calculation by the authors' formula is a lower variation from the county average than the averages of results calculated by the Parr formula.

These counties were selected at random and the results reported as determined. There may be counties which show the authors' formula to better advantage but if so our calculations did not discover them.

It appears to the writer that coals of the type of the Illinois coal with their characteristic high sulfur content afford the best possible test of the usefulness of formulas designed to accomplish correction in the ash value. Of proposed formulas to accomplish this correction the one that gives the most satisfactory results when applied to the analyses of these coals should be adopted or retained as the case may be in preference to others. The fact that the results obtained by the use of the authors' formula are less satisfactory than the results obtained by the use of the Parr formula seems to the writer to make advisable the retention of the Parr formula at least until another which is more satisfactory is developed.

A further question arises in regard to the formula proposed by the authors to express pure coal substance. Since the desire is to eliminate only the noncoal material it does not appear logical to eliminate the sulfur which is organic and hence actually part of the coal. Therefore, the noncoal material should be represented by the expression $M + 1.08A + 0.25S$. The complete formula would then be

$$\text{Unit Coal B.t.u.} = \frac{\text{Indicated B.t.u.} - 5247 \times \frac{1}{2}S}{1.00 - (M + 1.08A + 0.25S)}$$

A. C. Fieldner and W. A. Selvig.—The data which Dr. Cady has presented in his discussion of our paper are of the greatest importance in arriving at a decision on the most suitable formula for correcting ash back to the original mineral matter in the coal. The modified formula which we presented has a logical basis in that a study of forms of sulfur in various coal shows the percentage of pyritic sulfur to average in the neighborhood of one-half of the total sulfur. The Parr formula assumes practically all the sulfur to be present in the pyritic form. Cady's data clearly show that regardless of theory, the Parr formula actually shows a slightly smaller average variation and a smaller range of variation. Either the Illinois coals covered by this survey have partically all the sulfur in the pyritic form or certain other compensatory changes are inherent in the Parr formula.

It is recognized that the Illinois type coal affords a good test of the usefulness of formulas designed to correct ash values. Nevertheless it is hoped that the modified formula and the original Parr formula will be checked up against other coals as well as those of the Illinois type. Furthermore, Dr. Cady's suggestion about correcting only for pyritic sulfur and considering the organic sulfur a part of the coal material is worthy of consideration.

Determination of Mineral Matter in Coal and Fractionation Studies of Coal

By E. Stansfield* and J. W. Sutherland,† Edmonton, Alta.

(New York Meeting, February, 1930)

It is well known that the ash left when coal is burned is not the same either in chemical composition or in weight as the mineral matter originally present in the coal. This mineral matter has been referred to as "mineral impurity," "ash-forming constituents" and "corrected ash." The authors prefer the simple name used in the title and throughout this paper.

The mineral matter in a sample of coal cannot be isolated unchanged and then weighed; but if the relation between the weight of mineral matter in the coal and the weight of ash remaining when the coal is burned can be ascertained, the weight of mineral matter in the sample can be calculated from the determined weight of ash. When the weight of mineral matter in the sample is known, the weight of pure coal can be found by subtraction. This pure coal must be the basis of accurate coal classification.

W. F. Wheeler,[1,3] W. Brinsmaid,[2] S. W. Parr,[3] and many subsequent workers have studied the relation between mineral matter and ash. Wheeler and Parr and others have deduced formulas for calculating this ratio, and used heavy-solution separations for checking the accuracy of their formulas. Brinsmaid hand-picked and analyzed two separate samples (and mixtures of these) and based thereon a graphical method of deducing the ratio. In the Fuel Testing Laboratories of the Scientific and Industrial Research Council of Alberta, a graphical method, based on heavy-solution fractionation of coal samples, has been used with marked success for the past 7½ years for determining the mineral matter-ash ratio. This method resembles that employed by Brinsmaid in 1907.

---

* Chief Chemical Engineer, Scientific and Industrial Research Council of Alberta.

† Fuel Analyst, Scientific and Industrial Research Council of Alberta.

[1] W. F. Wheeler: Pure Coal as a Basis for the Comparison of Bituminous Coals. *Trans.* A. I. M. E. (1907) **38**, 621.

[2] W. Brinsmaid: The Amount of Inert Volatile Matter in the Mineral Constituents of Coal. *Jnl. Ind. & Eng. Chem.* (1909) **1**, 65.

[3] S. W. Parr and W. F. Wheeler: Unit Coal and the Composition of Coal Ash. Univ. Ill. Eng. Expt. Sta. *Bull.* 37 (1909).

Coal Fractionation by Heavy Solutions

Many forms of apparatus have been devised to expedite the work of separation of float-and-sink samples with heavy solutions. The method used in this laboratory employs centrifuge cups of the type shown in Fig. 1. These consist of brass cylinders, 6¼ in. long and 2 in. dia., closed at the bottom, fitted with a cone constriction in the middle and a screw-operated cone valve. The valve can be readily removed by screwing off the cap. The slope of the sides of the cones was made 60° to

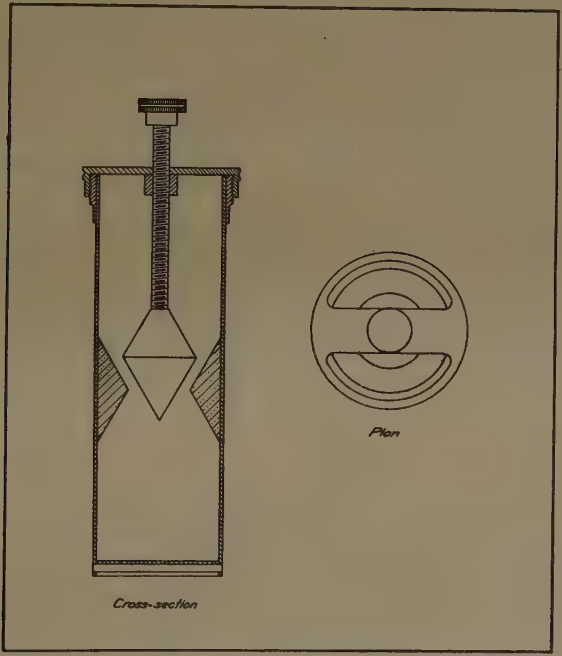


Fig. 1.—Centrifuge cup.

insure against particles being retained on their surface and to facilitate removal of the treated sample. The heavy solutions employed are made by mixing carbon tetrachloride and benzene. Four different gravities are used with each coal; these must be adjusted to suit the particular coal studied, but typical values for Alberta coals are 1.30, 1.33, 1.35 and 1.38.

The procedure is as follows: A representative 200-g. sample of the coal to be tested is further ground in a mill to pass a 28-mesh Tyler standard screen. The sample is then partly dried for about ½ hr. in a current of natural gas at 105° C. and cooled in an exhausted desiccator. Natural gas, or carbon dioxide, is used, as good results can not be obtained with lower rank coals unless precautions against oxidation are taken

during certain stages of the work. A 30-g. portion of the coal is weighed into a centrifuge cup, some of the desired heavy solution added, and the charge shaken or stirred. The valve is then inserted into the cup, but not closed, and solution added to nearly fill the cup. Two or four such cups are prepared with solutions of different gravity and their weights equalized by careful additions of the correct solution. The cups are then centrifuged for 10 min. at 1500 r.p.m. in a size 2 centrifuge made by the International Equipment Co. The valves are then closed and the floats poured out on to a filter paper in a Buchner funnel. A brush can be used to clean out coal that adheres to the cup. Suction for a few minutes on the filter removes most of the solution, but the removal of final traces can be insured by slight warming in a vacuum oven or even by standing in an evacuated desiccator. The coal is transferred as completely as possible from the filter to a suitable container and weighed after complete removal of solution. After removal of the floats from the cups, the valves are taken out and the sinks similarly collected, dried and weighed. The eight fractions thus obtained, and a sample of the original coal, are finely ground in small porcelain ball mills, from which the air is displaced by natural gas. The ground samples are preserved in tightly stoppered bottles, but the determinations of moisture, ash and calorific value are made with the least possible delay. Other analyses are sometimes made, but those described are the essential ones for the study.

Study of Results

The calorific values and ash percentages are calculated to a dry-coal basis and plotted as shown in Fig. 2. These points, for all Alberta coals so far tested, are found to lie, within experimental errors, along a straight line AB. This line produced to the left cuts the line of zero ash at the point C, 13,000 B.t.u. in the figure. It is obvious that this may reasonably be assumed to be the calorific value of the pure dry coal.

If any pure coal of calorific value H_0 is mixed with an inert material so that the percentage of inert material—that is, mineral matter—in the mixture is M, then the calorific value of the mixture Hm is given by the equation $Hm = H_0 \frac{100 - M}{100}$. Or the percentage of mineral matter in the mixture of calorific value Hm is given by the equation $M = \frac{100(H_0 - Hm)}{H_0}$. Thus in Fig. 2, coal of calorific value D contains K per cent. mineral matter as represented by the point E. Coal of calorific value D, however, was found to give an ash yield of G per cent., as represented by the point B. The ratio of K to G, or of the line DE to DB, is the ratio of mineral matter to ash in this coal fraction. Points

similar to E could be found for each of the other coal fractions, and it is clear that these lie on the line CP; that is, on the line joining the point of calorific value of the pure coal with the point of zero calorific value and 100 per cent. mineral matter. If the line AB is produced to the right, it cuts the line of zero calorific value at the point L. And the mean ratio of mineral matter to ash for all the coal fractions is given by the ratio FP to FL; that is, by 100 to 80, or 1.25 in the figure.

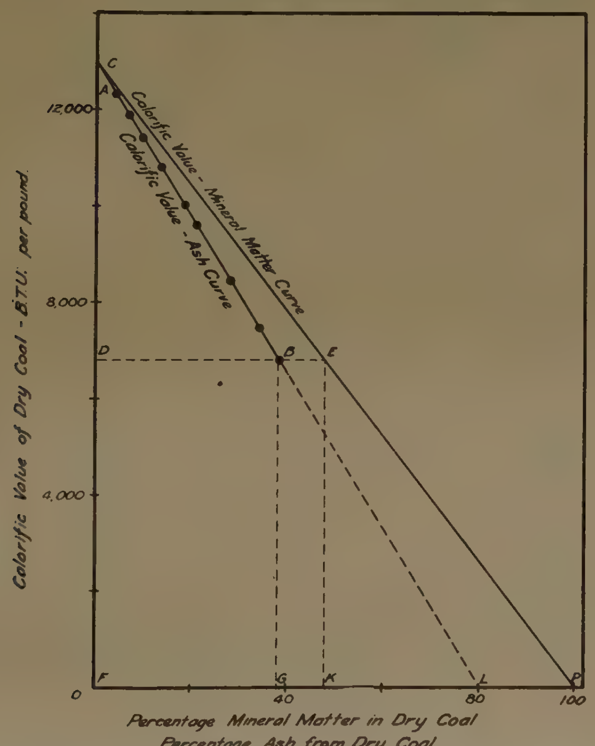


FIG. 2.—CALORIFIC VALUES AND ASH PERCENTAGES CALCULATED TO A DRY-COAL BASIS.

The ratio FP to FL is the same as that of DE to DB, if the point B lies exactly on the line CL, but the significance of the point L is clearer. A sample containing no coal—*i. e.*, 100 per cent. mineral matter—on ignition leaves (in the case shown) 80 per cent. ash.

Any coal sample may contain more than one type of pure coal, with varying calorific values, and may contain more than one mineral impurity, with varying mineral matter-ash ratios. These constituents may be separated in varying proportions in the fractionation method employed, so that there was no reason to anticipate that the calorific value-ash curve AB would be a straight line. The only exception noted to a

straight-line relation with Alberta coals has been a barely noticeable tendency for very clean samples to have slightly lower calorific value than anticipated. The preparation of such clean samples, however, involves the discard of a large fraction of the pure coal substance. If, however, a straight-line relation were not found, the mineral matter-ash ratio could be calculated for each fraction, as already indicated. If this ratio varied with the purity of the coal, a ratio-ash curve could be plotted. It is necessary, however, in every case to deduce a calorific value of the pure coal by extrapolation.

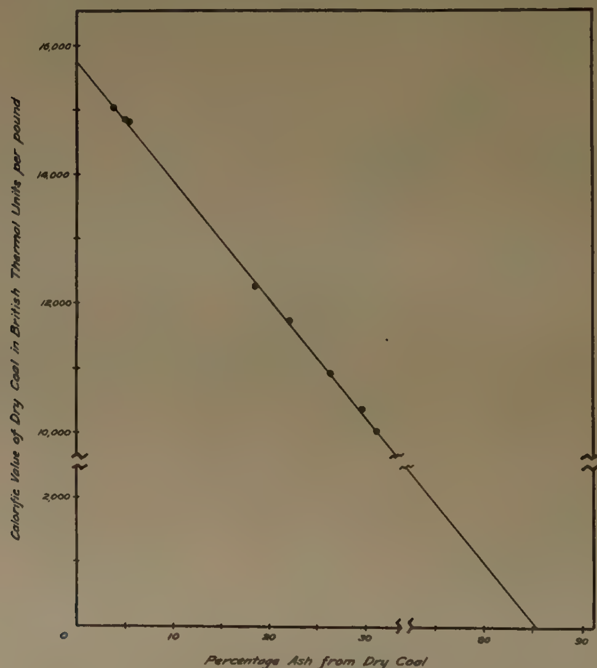


FIG. 3.—ASH-CALORIFIC VALUE CURVE.

Table 1 shows the actual values found with a sample of bituminous coal from the Mountain Park area in Alberta. Fig. 3 shows these results graphically. The zero-ash coal has a calorific value of 15,770 B.t.u. and the ash percentage corresponding to zero calorific value is 85.5. This gives the ratio of mineral matter to ash as 1.17. In Table 1 the ratios obtained by calculation vary from 1.130 to 1.188. It is clear that even small experimental errors will give large errors in the ratios calculated for low-ash fractions. For the last five fractions the ratio only varies from 1.16 to 1.19. The weighted average for the calculated ratios is also 1.17.

Coals from more than 30 mines have been examined and their ratios found to range from 1.03 to 1.28, but the values usually lie within the

limits 1.08 to 1.19. It should be particularly noted that the above ratio is an all-embracing value allowing for presence of iron pyrites, carbonates, hydrates, etc. It must not be confused with the ratio for which Parr assumed an average value of 1.08, as Parr made a separate adjustment for iron pyrites.

TABLE 1.—*Analyses of Fractions of Bituminous Coal*

Sample from Mine in Mountain Park Area, Alberta

Sample	Analysis of Moist Coal			Analysis Dry Coal		Mineral Matter by Cal.,[a] Per Cent.	Ratio Mineral Matter[b] to Ash
	Moisture, Per Cent.	Ash, Per Cent.	Cal. Val., B.t.u. per Lb.	Ash, Per Cent.	Cal. Val., Per Cent.		
Float on:							
1.30 sp. gr. sol.	0.53	3.93	14,954	3.95	15,033	4.67	1.183
1.37 sp. gr. sol.	0.47	5.06	14,752	5.08	14,822	6.01	1.183
1.39 sp. gr. sol.	0.49	5.39	14,731	5.42	14,804	6.12	1.130
1.42 sp. gr. sol.		5.57					
Original sample	0.68	18.46	12,201	18.60	12,285	22.10	1.188
Sink in:							
1.30 sp. gr. sol.	0.31	21.98	11,686	22.05	11,717	25.71	1.166
1.37 sp. gr. sol.	0.40	26.31	10,865	26.42	10,909	30.82	1.167
1.39 sp. gr. sol.	0.65	29.26	10,309	29.46	10,376	34.18	1.160
1.42 sp. gr. sol.	0.51	30.79	9,970	30.95	10,021	36.46	1.178

[a] Calorific value of pure, dry coal, as found by graph, 15,770 B.t.u. per pound.
[b] Ratio of mineral matter to ash, as found by graph, 1.17.

CALCULATION OF CALORIFIC VALUE OF PURE COAL

When the ratio of mineral matter to ash f has been determined for a coal, the calorific value H_0 of the pure, or mineral-matter-free, coal can be calculated from the calorific value H_A of any sample with A per cent. of ash by the equation:

$$H_0 = H_A \frac{100}{100 - fA} \qquad [1]$$

The usual equation employed for calculating the calorific value of the so-called ash-free coal, neglecting the difference between mineral matter and ash, is:

$$H_0 = H_A \frac{100}{100 - A} \qquad [2]$$

The error introduced by the use of the approximate equation 2 increases with the value of f and increases with the ash yield of the sample.

The results obtained by equation 2 can be obtained graphically on such a chart as Fig. 2 by drawing a series of lines from the point P through the different points from A to B, and continuing these lines to cut the axis CF, that is the zero-ash axis. Each line cuts the axis at a different calorific value, and in each case below the correct, pure-coal calorific value; the higher the ash, the greater the error.

The error introduced by the use of equation 2 is shown graphically in Fig. 4, for samples up to 20 per cent. ash, and for ratios f from 1.0 to 1.3. These curves show that if, for example, the calorific value of a coal sample of 10 per cent. ash is used to calculate the calorific value of pure coal, neglecting to correct for the mineral matter to ash ratio, the result may be as much as 3 per cent. too low.

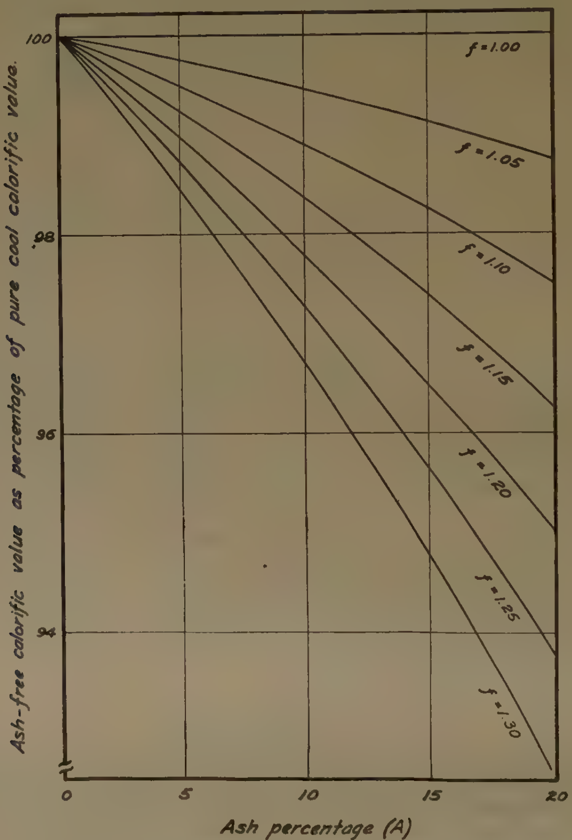


FIG. 4.—CALORIFIC VALUE OF ASH-FREE COAL.

Calculated from coal of ash percentage (A) and expressed as percentage of true calorific value of pure coal. Curves for seven values of ratio of mineral matter to ash (f).

It should be noted that in the above discussions all the samples are assumed to be dry. To calculate the calorific value H_0 of pure dry coal from the calorific value H_{MA} of a sample with M per cent. moisture and A per cent. ash, equation 1 has to be modified to:

$$H_0 = H_{MA} \frac{100}{100 - (M + fA)}$$

Significance of Mineral Matter to Ash Ratio

The three principal mineral-matter constituents which notably change in weight when burned to ash are (1) carbonates, (2) pyrites, and (3) shales containing water of hydration. W. P. Campbell[4] studied three coals to ascertain how far the proved presence of these constituents would go towards accounting for the difference between the calculated mineral matter and the determined ash. He determined the carbonates in the coal and in the ash, and also determined sulfur forms and sulfur content in the coal and in the ash of a heavy fraction of each of the coals. From his results he calculated the loss in weight of the mineral matter, due to loss of carbon dioxide and combustion of pyrites. No method was known for determining the water of hydration of the shales, so that it was only possible to assume that this constituted the otherwise unaccounted-for loss. His results are summarized in Table 2. With

Table 2.—*Campbell's Results*

	Coal A	Coal B	Coal C
Calorific value pure, dry coal from graph, B.t.u.	13,930	15,950	12,740
Ratio of mineral matter to ash, from graph	1.27	1.11	1.05
Test fraction:			
Calorific value, determined, B.t.u.	10,809	10,926	9,712
Mineral matter content, calculated, per cent	22.4	31.6	23.75
Ash yield, determined, per cent	18.1	28.5	22.7
Loss of mineral matter, by difference, per cent	4.3	3.1	1.05
Loss due to carbonates, per cent	3.25	0.15	0.40
Loss due to pyrites, etc., per cent	0.00	0.29	0.00
Loss due to water of hydration (by difference), per cent	1.05	2.66	0.65
Ultimate analyses of fractions:			
Ash yield, float fraction, per cent	3.3	2.1	
Ash yield, sink fraction, per cent	31.2	40.5	
Carbon-hydrogen ratio, float fraction	19.1	21.8	
Carbon-hydrogen ratio, sink fraction	20.9	18.3	

coal A there is a large loss due to carbonates and a smaller loss due to shales (water of hydration). With coal B there is practically no loss due to carbonates, but a large loss due to shale. With coal C all the losses are small. Sulfur, whether as pyrites or as sulfates, is negligible for coals A and C and has only a slight effect with B.

Campbell obtained further confirmation of this interpretation of his results by comparing the ultimate analyses of low-ash and high-ash

[4] W. P. Campbell: Master's thesis, University of Alberta, 1925.

fractions of coals A and B. With coal A, where the mineral matter is high in carbonates and contains little water of hydration of shale, the carbon-hydrogen ratio rose with the ash. With coal B, on the contrary, where carbonates were low but the water of hydration of shale high, the carbon-hydrogen ratio fell with rising ash.

Variations of Mineral Matter to Ash Ratio in a Seam within the Limits of a Mine

Five separate samples of coal from the same seam in a bituminous mine in the Crow's Nest area of Alberta were available for study. Four of these were channel samples across the seam, and the other was a ½-

Table 3.—*Analyses of Bituminous Coal from Mine in Crow's Nest Area, Alberta*

Sample	Analysis of Dry Coal		Mineral Matter by Calculation,[a] Per Cent.	Ratio Mineral Matter to Ash[b]
	Ash, Per Cent.	Cal. Val., B.t.u. per Lb.		
11A–29 float on 1.54 sp. gr. sol.	6.20	14,396	7.48	1.21
11–29 channel sample	13.05	13,254	14.81	1.13
10–29 channel sample	13.16	13,293	14.56	1.11
12–29 channel sample	18.94	12,114	22.13	1.17
9–29 channel sample	25.14	14,024	29.13	1.16
404–29 Half-ton consignment:				
Selected lump	4.63	14,720	5.40	1.17
Selected lump	5.00	14,623	6.02	1.20
Selected lump	8.09	14,074	9.56	1.18
Selected lump	8.84	13,843	11.03	1.25
Selected lump	13.90	12,280	21.08	1.52
Selected lump	14.01	13,110	15.74	1.12
Selected lump	14.20	13,000	16.45	1.16
Selected lump	14.47	12,955	16.74	1.16
Selected lump	15.39	12,749	18.07	1.17
Average sample	17.17	12,487	19.75	1.15
Selected lump	19.88	11,270	27.6	1.39
Selected lump	21.00	10,290	33.9	1.61
Selected lump	21.83	11,795	24.2	1.11
Selected lump	29.37	10,250	34.1	1.16
Selected small pieces—about 30	33.7	9,570	38.5	1.14
Selected lump	39.5	8,463	45.6	1.15
Selected small pieces—about 30	39.6	7,870	49.4	1.25
Selected lump	39.7	9,189	40.9	1.03
Selected lump	49.1	6,870	55.8	1.14
Selected lump	75.5	2,941	81.1	1.07

[a] Calorific value of pure dry coal assumed, as from graph, 15,560 B.t.u. per pound.
[b] Mineral matter to ash ratio, from graph, 1.15.

ton consignment. One of the channel samples was floated on a 1.54 specific gravity solution to obtain a cleaner sample, and 19 selected samples were taken from the ½-ton lot, in addition to the regular sample. Of the selected samples, 17 were single lumps and 2 were made up of small pieces. There were thus obtained 25 separate samples giving ash yields ranging from about 5 to 75 per cent. When the calorific values and ash yields of these samples were plotted it was found that although the majority fell with reasonable accuracy along a straight line, there were some notable exceptions. If the calorific value of the pure, dry coal is assumed to be 15,560 B.t.u. (as shown by the line on the graph) for all the samples—an assumption, however, which is far from justified—the ratios are as shown in Table 3. The average sample and channel samples show reasonable agreement but there are some marked exceptions with the single lumps. It may be noted that pillar coal was being worked in one section of the mine at the time, so that some of the coal was probably weathered, and that a study of the fusibilities of the ash samples showed that the nature of the ash was also markedly variable.

Oxidation and Weathering Studies of Coal

The oxidizability of any coal can be studied by means of the calorific value-ash curves already described. The test developed in these laboratories is as follows. One-half of a sample of fresh coal, ground to just pass a 28-mesh screen, is fractionated as before, and a calorific value-ash curve is obtained. The other half of the sample is exposed for 6 hr. in a current of air in a toluol oven at 106° C. Float, sink and original samples of the oxidized coal are then analyzed and the points placed on the same graph, as shown at *D*, *E* and *F* in Fig. 5. The reasonable assumption is made that the nature of the mineral matter is not changed

Table 4.—*Comparative Oxidation Values for Eleven Alberta Coals*

Sample	Analysis of Coal as Mined					Calorific Value Pure, Dry Coal, B.t.u. per Lb.	Reduction of Calorific Value on Oxidation for 6 Hr. at 106° C.	
	Moisture, Per Cent.	Ash, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Cal. Val., B.t.u. per Lb.		B.t.u. per Lb.	Per Cent.
69–27	0.9	7.5	15.8	75.8	14,270	15,700	60	0.4
74–27	2.1	6.7	21.8	69.4	14,300	15,920	130	0.8
87–26	3.6	10.0	22.8	63.6	13,350	15,750	175	1.1
78–27	1.1	9.6	30.4	58.9	13,800	15,560	130	0.8
53–27	7.2	13.9	34.9	44.0	11,470	14,880	240	1.6
86–26	10.6	8.3	35.7	45.4	11,270	14,190	360	2.5
94–26	14.3	11.4	30.4	43.9	10,140	13,900	370	2.7
56–27	16.0	8.8	29.1	46.1	9,860	13,210	270	2.0
3–27	17.9	3.5	31.4	47.2	10,420	13,420	520	3.9
22–27	19.5	6.2	30.7	43.6	9,710	13,300	410	3.1
58–27	25.8	4.9	29.2	40.1	8,870	13,010	350	2.7

during oxidation, so that the calorific value-ash curve of the oxidized coal must meet that of the fresh coal on the zero calorific-value axis. This in general practice is found to be the case. The line from the point C through F, E and D cuts the pure-coal axis at the point G below the point A of the fresh coal. This drop in calorific value of the pure coal is a measure of the comparative oxidizability of the coal and can be

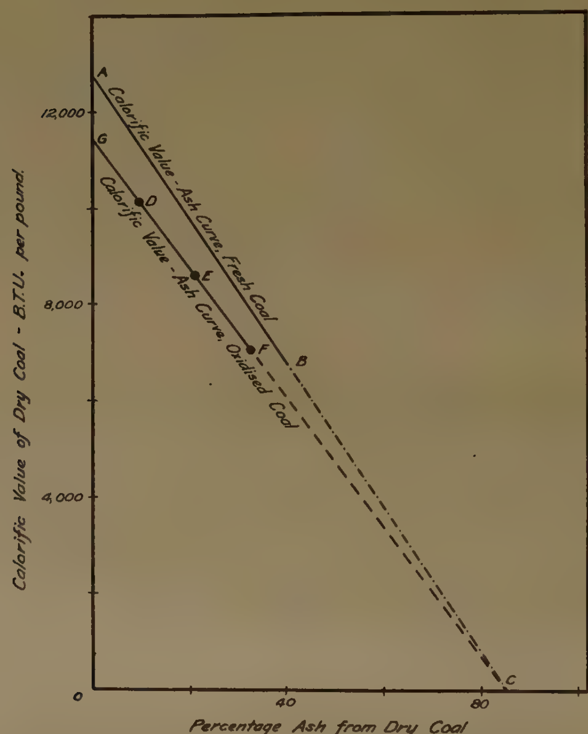


FIG. 5.—COMPARISON OF FRESH AND OXIDIZED COAL.

expressed either in British thermal units or preferably as a percentage of the calorific value of the pure, fresh coal.

Table 4 shows the comparative oxidation values for 11 typical Alberta coals, tested by the method described above. The oxidizability of the coal tends to increase with the lowering rank of the coals.

The degree of weathering of any sample of coal can be judged by plotting the dry-basis calorific value-ash point of the sample on a chart showing the normal calorific value-ash curve of fresh coal from the same seam and mine. This method of study has been found to be particularly useful in storage tests of coal, where the samples are not otherwise readily comparable on account of varying ash.

Fractionation Curves for Constituents Other than Calorific Value

This discussion has been confined to the relations between calorific value, mineral matter and ash. Other analyses of the separated fractions of coal have been made and studied. Carbon-ash curves show

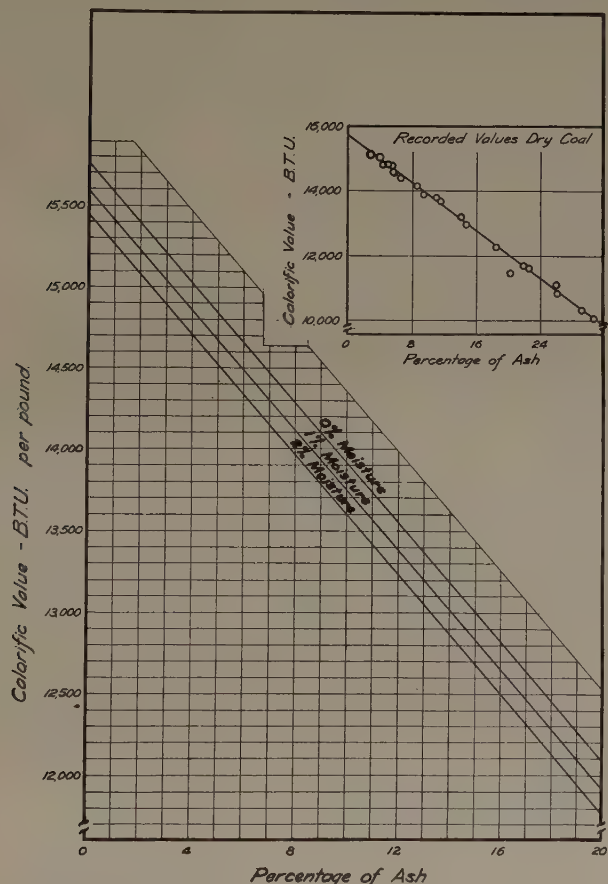


Fig. 6.—Commercial calorific value-ash curves.

regularity and the method is applicable for the evaluation of carbon in the pure coal. As the carbon from the carbonates in mineral matter affects the determined carbon content, an erroneous mineral matter-ash ratio is shown. Hydrogen-ash curves are similarly affected by the water of hydration of the shales; and the percentage error on such a low value as that of hydrogen is too great for accuracy. No satisfactory volatile matter-ash curves or moisture-ash curves have yet been plotted. Further work should be done along the above lines.

Commercial Adaptation of Calorific Value-ash Curves

Fig. 6, which is self-explanatory, illustrates the type of chart supplied to coal operators in Alberta. Three curves are here given, for coal with zero, 1 per cent. and 2 per cent. moisture respectively. Calorific values for samples with intermediate moisture contents can be found by interpolation; coal from the mine in question usually has between 1 and 2 per cent. of moisture. It should be noted that both the calorific value and the ash in these curves are for coal with the moisture shown.

Inset in the upper corner is a chart on a smaller scale, showing for dry coal only the actual points found with coal from that mine.

DISCUSSION

H. J. Rose, Pittsburgh, Pa.—The arithmetical calculation of B.t.u. to the basis of ash-free coal substance gives too low a value. This discrepancy may amount to hundreds or even a thousand or more B.t.u., in the case of high-ash coals. This can be graphically demonstrated on Fig. 2. The graphic equivalent of the ordinary method of calculating coal B to the ash-free basis would be to draw a line from 100 per cent. mineral matter (point P) through B until the line intersects the left boundary of the chart, which it would do at 11,000 B.t.u. (or about 2000 B.t.u. below the correct value as indicated by the authors' method or by comparison with the B.t.u. of the low-ash samples).

The authors have presented a workable and convincing method which merits the consideration of everyone who is interested in calculating B.t.u. from ash analysis results. I use a simpler method based on the same principle, which consists in simply plotting B.t.u. vs. ash on ordinary cross-section paper. B.t.u. values for ash contents other than those already determined can be obtained by interpolation or extrapolation, although, of course, it is not feasible to extrapolate to the extent that is possible when sink-and-float samples are included on the graph, as recommended by the authors.

Constitution and Nature of Pennsylvania Anthracite with Comparisons to Bituminous Coal*

By Homer Griffield Turner,† Bethlehem, Pa.

(New York Meeting, February, 1930)

The nature and comparative features of anthracite and bituminous coals have been discussed by the writer in two previous papers.[1] Although this paper is offered as a further contribution to the subject, it seems desirable at the outset to repeat some of the descriptive matter of the early works in order to show the actual connection between various constituents of both coals and the new photomicrographs and other physical and chemical properties.

Structure of Anthracite and Bituminous Coals

A lump of bituminous coal (Fig. 1) is readily seen to be composed of layers differing from one another in texture, luster and thickness. There are, in general, three kinds of layers, although only two are usually seen. One is deep black, very compact, and has a bright luster; another is grayish black, has a duller luster and is less compact; the third is black, dull and porous in appearance, but because of its thinness is not often seen except on parting surfaces. This last material is charcoal.

American students of coal morphology use the names anthraxylon, attritus, and fusain to designate these three constituents of laminated coal. Anthraxylon refers to the brightest bands, attritus to the duller layers, and fusain to the charcoal.

Close examination of a lump of bituminous coal shows the brightest deep black layers to be embedded in the duller grayish black laminae. These deep black layers are lens-shaped and vary in thickness from thin sheets to over half an inch. They are often so continuous through the lump that their tapering terminae are not noticeable. On careful examination the duller grayish black laminae (Fig. 7) are seen to contain many very thin sheets of this very bright black material. The charcoal

* Presented before the Division of Industrial and Engineering Chemistry, American Chemical Society, Columbus Meeting, April, 1929.

† Assistant Professor of Geology, Lehigh University.

[1] H. G. Turner and H. R. Randall: A Preliminary Report on the Microscopy of Anthracite Coal. *Jnl. Geol.* (1923) **31**, 306.

H. G. Turner: Microscopical Structure of Anthracite. *Trans.* A. I. M. E. (1925) **71**, 127.

1 2 3 4 5 6

FIGS. 1–6.—CAPTIONS ON OPPOSITE PAGE.

when seen in vertical sections is in lenses or angular chips. Its presence is readily recognized on the parting faces parallel to the bedding planes (Fig. 8).

The shape of a lump of bituminous coal is roughly rectangular, due to the fact that the joints are vertical and the horizontal laminae not closely knit.

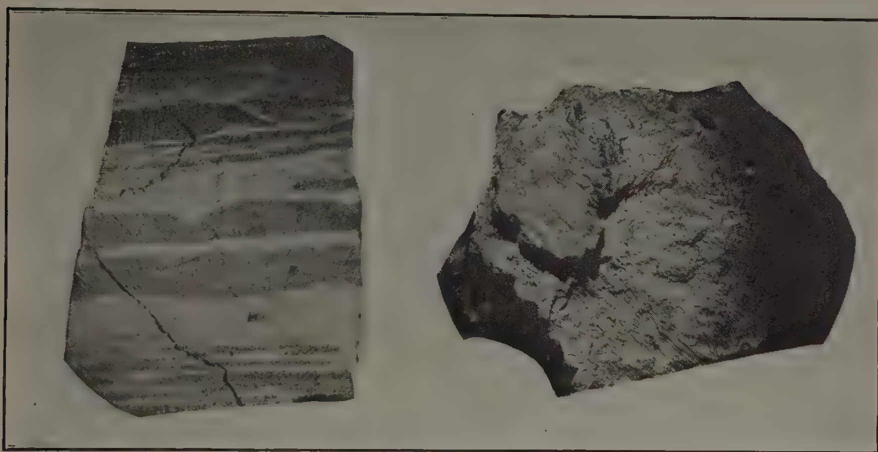
7

8

Fig. 7.—Lump of anthracite, ground to plane and etched by placing on hot iron plate. Natural size.

Note lens shape of dark woody bands. Note also that duller coal in lower half inch of figure contains many small lenses of woody tissue.

Fig. 8.—Parting face of bituminous coal, showing charcoal chips. Natural size.

On the whole, a lump of anthracite (Fig. 2) shows all the foregoing structural characteristics of bituminous coal with the exception of the shape of the lump, which is usually not so rectangular. There are a few exceptional cases (Fig. 4) where anthracite is so jointed and loosely laminated that their fractured fragments are fully as rectangular as

---

Fig. 1.—Lump of bituminous coal, showing tendency to break into rectangular fragments. Natural size.

Dark bands are woody laminae and lighter colored bands represent duller laminae, consisting of many kinds of plant debris together with mineral matter.

Fig. 2.—Lump of anthracite showing more conchoidal fracture than bituminous coal. Natural size.

Laminations are not so distinct as in bituminous coal.

Fig. 3.—Rear view of Fig. 2. Natural size.

This face has been ground to a plane surface and placed on an iron plate, which was raised to a red heat. Resulting etching has emphasized dark woody laminae and lighter attritus.

Fig. 4.—Lump of anthracite. Natural size.

Note similarity to bituminous coal in laminae and shape of fragment.

Fig. 5.—Lump of anthracite, showing high conchoidal fracture. Laminated character barely noticeable. Natural size.

Fig. 6.—Rear view of Fig. 5 deeply etched on hot iron plate to show laminations. Natural size.

Note scarcity of dark bands of woody matter.

those of bituminous coal. This difference in the shape of anthracite lumps is due to fewer vertical fractures and to greater cohesion between laminae, giving anthracite a more pronounced conchoidal fracture.

The luster of anthracite is brighter and more uniform than that of bituminous coal, so that its laminated character is not always so readily seen. The bright layers of bituminous coal are brilliant layers in anthracite; the duller layers of bituminous are bright layers in anthracite; while the charcoal in both coals is very much alike (Fig. 9).

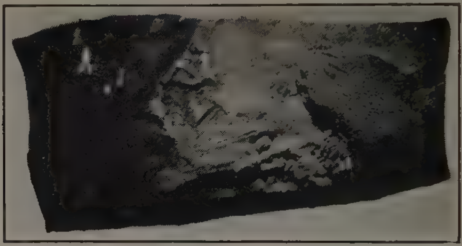


FIG. 9.—PARTING FACE OF ANTHRACITE, SHOWING CHARCOAL CHIPS. NATURAL SIZE.

Some beds of anthracite and some parts of other beds are so compact and so uniformly bright that the whole mass seems to lack laminations. This type (Fig. 5) of anthracite breaks with a high conchoidal fracture and throws off chips which are sharp enough to cut the hands. As a matter of fact, the laminae (Fig. 6) in these cases are far below the average in number and thickness. The fact that the thickest and most widespread bed in the Pennsylvania anthracite region is largely of this nature has given rise to the statement so often found in the literature that anthracite is not laminated. With the exception of much of this bed and a few restricted portions of other beds, all Pennsylvania anthracite is as fully laminated as bituminous coal. Figs. 3 and 4 show this very clearly.

FIG. 10.—BRIGHTEST LAMINAE, SHOWING ALMOST PERFECT WOOD CELLS. × 150
Note intercellular spaces indicated by small dots.

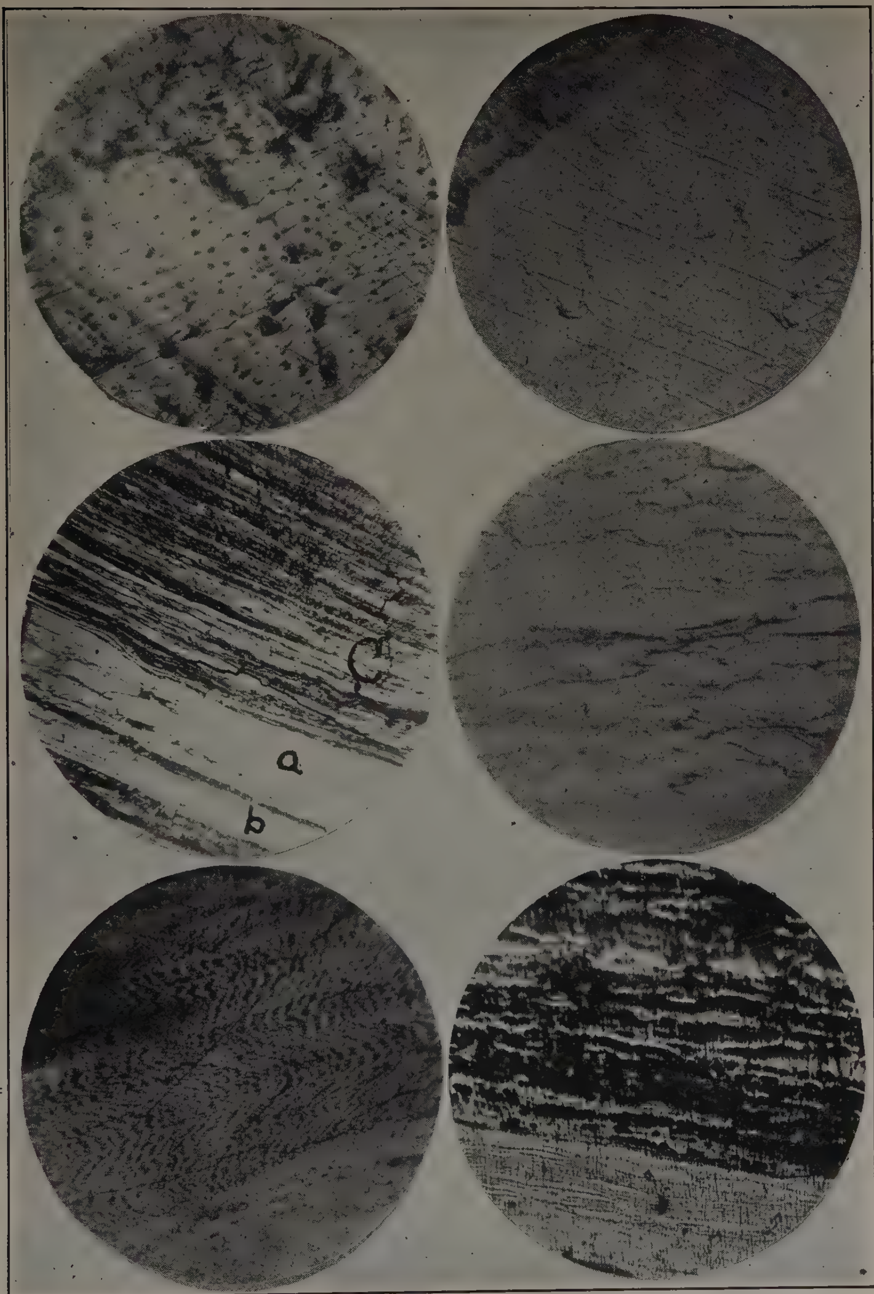
FIG. 11.—STRIP OF WOOD WITH CELLS SO COMPRESSED AS TO BE BARELY NOTICEABLE. × 150.
Cells are indicated by dark broken lines at upper left side of photograph.

FIG. 12.—AN AREA OF ANTHRACITE. × 13.
a, *b*, Thick bands of wood; *c*, Typical view of duller laminae, containing in this case, many thin strips of wood, small spores, and other plant fragments.

FIG. 13.—WOODY BAND IN *b* IN FIG. 12. × 150.
This highly magnified view shows that cells of wood were filled with resinous matter. This type of tissue is very common in all coals examined. Many former resinous lumps of this kind are found detached in duller laminae.

FIG. 14.—ONE OF BRIGHTEST BANDS IN ANTHRACITE SHOWING WELL PRESERVED WOOD STRUCTURE. × 34.

FIG. 15.—POLISHED SURFACE OF BITUMINOUS COAL ETCHED BY CHROMIC ACID. × 78.
Broad light-colored band is a strip of wood, showing cell structure; darker area duller lamina composed of spores and plant debris. Compare with Fig. 12.



10 11

12 13

14 15

FIGS. 10–15.—CAPTIONS ON OPPOSITE PAGE.

Appearance Under Microscope

The character of the various laminae of both anthracite and bituminous coal is brought out clearly under the microscope. In this paper, the details of anthracite, which is opaque in thin sections, are revealed through the use of polished surfaces which have been etched by means of the blowpipe flame.[2] The details of bituminous coal are shown by means of thin sections and by polished surfaces etched by Seyler's[3] chromic acid method. Only enough photomicrographs of bituminous coal are used to clearly establish the relationship between the two ranks of coal. More complete descriptions of bituminous coal can be found in the writings of David White and Reinhardt Thiessen.[4]

The Brightest Laminae or Anthraxylon

The brightest bands, so prominent in both anthracite and bituminous coal, are found to be pieces of wood, such as limbs, stems, twigs and

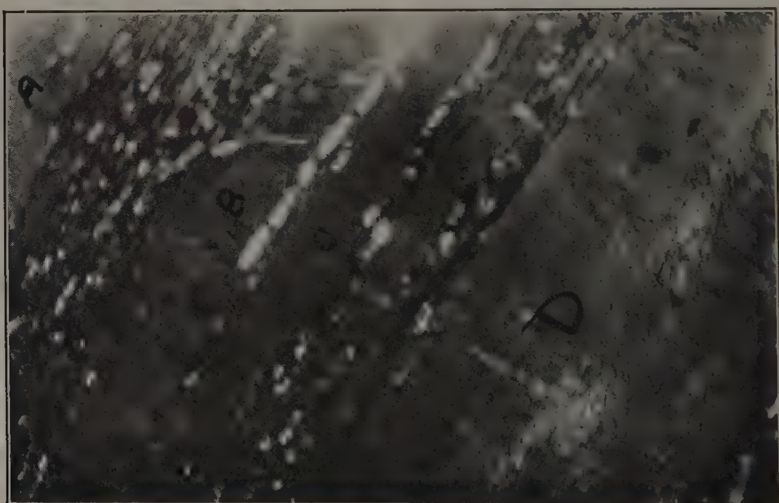


Fig. 16.—Thin sections of bituminous coal, showing woody strips, *a*, *b*, *c*. × 200. Largest strip *d*, shows well defined structure. Areas between wood represent duller laminae composed of plant debris. Light disks in these layers are spores and resinous matter.

roots. The wood cells range from almost perfect forms (Fig. 10) to forms that have been greatly flattened (Fig. 11). Most of these woody

[2] H. G. Turner: Microscopical Structure of Anthracite. *Trans.* A. I. M. E. (1925) **71,** 127.

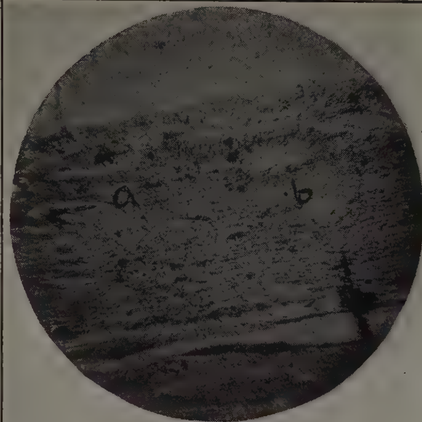
[3] C. A. Seyler: The Microstructure of Coal. *Trans.* A. I. M. E. (1925) **71,** 117.

[4] D. White and R. Thiessen: Origin of Coal. U. S. Bur. Mines *Bull.* 38 (1914).
R. Thiessen: Structure in Paleozoic Bituminous Coals. U. S. Bur. Mines *Bull.* 117 (1920).

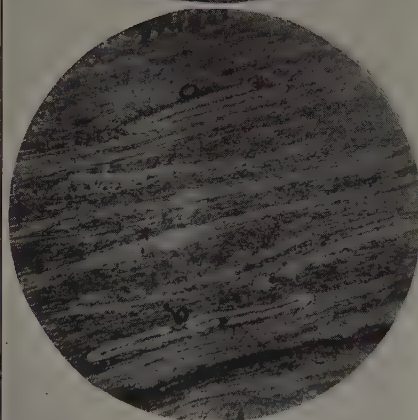
fragments seem to have been compressed. In anthracite and in bituminous coal this compression has caused some of the wood to appear almost structureless. Figs. 10, 11, 12, 13 and 14 show various views of the woody components of anthracite while Figs. 15 and 16 show comparable forms in bituminous coal.

The Duller Laminae or Attritus

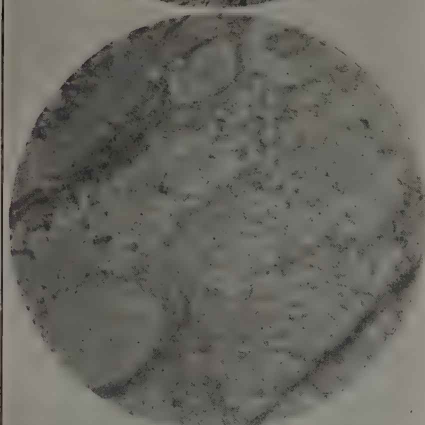
In anthracite and bituminous coals the duller laminae are seen to be composed of a great number of constituents derived from various parts of plants together with inorganic particles and a substance with the appearance of mud. In this mass are found spores, cuticle, lumps of resin, pith, plant-fibers, plant cells and parts of cells, finely macerated woody matter, carbonized material, and inorganic particles. These constituents are shown in Figs. 17, 18 and 19. These duller layers in coal are really the groundmass in which the woody lenses are embedded. Their duller luster is due partly to their heterogeneous constitution and partly to the earthy mineral matter and



17



18



19

Fig. 17.—Duller laminae of anthracite. × 13.
Note large spore at *a*, and formet resin lump at *b*. Thin strips represenr woody matter and probably leaf cuticle.

Fig. 18.—Details of duller laminae of anthracite. × 34.
Two large spores are shown at *a* and *b*. Thin strips are woody matter, leaf cuticle and flattened spores. Very small light spots are microspores.

Fig. 19.—A badly macerated piece of wood in anthracite, showing former circular resinous bodies. × 150.

charcoal which they contain. Very thin sheets of woody matter are usually prominent in these duller layers.

The Charcoal or Fusain

Charcoal is abundant in anthracite and bituminous coal. While most of it occurs in layers too thin to be seen with the naked eye except along the parting faces, some fairly large lenses are always visible in a hand specimen of coal. Layers 3 in. thick and 10 in. or more in diameter have been found in both coals. Masses of this size have the appearance of an aggregate of crisscrossing small chips of wood, each chip being about ¾ in. long, ½ in. wide and ¼ in. thick. The charcoal content of anthracite and bituminous coal is usually soft and brittle although at times it is quite hard, due to an infiltration of mineral matter or organic products produced during coal metamorphism. It shows the most perfect plant structure of all the coal constituents. The wood cells of which it is commonly composed are devoid of filling as a rule, and are so well preserved as to be clearly visible on polished surfaces of coal without the use of any etching agents. It often happens that these cells have been fractured and pushed together, but even in these cases, their identity is not lost. Figs. 20, 21, 22, 23, 24 and 25 represent charcoal in anthracite.

Chemical Composition of Laminae

With the purpose of showing the difference in chemical as well as morphological properties, the three distinct laminae in anthracite and bituminous coals were isolated and analyzed separately. The coals selected were the Forge Split of the Mammoth, an anthracite from Nanticoke, Pa., and the Freeport bed, a bituminous coal from Indianola, Ind. The results are given in Table 1.

Fig. 20.—Charcoal in anthracite, showing well preserved empty cells. × 150.

Small intercellular spaces can also be seen. Border cells have been fractured and pushed together.

Fig. 21.—Charcoal in anthracite. × 150.

a, Almost perfect cells; *b*, fractured cells.

Fig. 22.—Typical view of charcoal fragments in anthracite. × 34.

Fig. 23.—Charcoal in anthracite. × 150.

This carbonized piece has been fractured and part *B* has moved down with respect to *A*. Note fractured cells around borders.

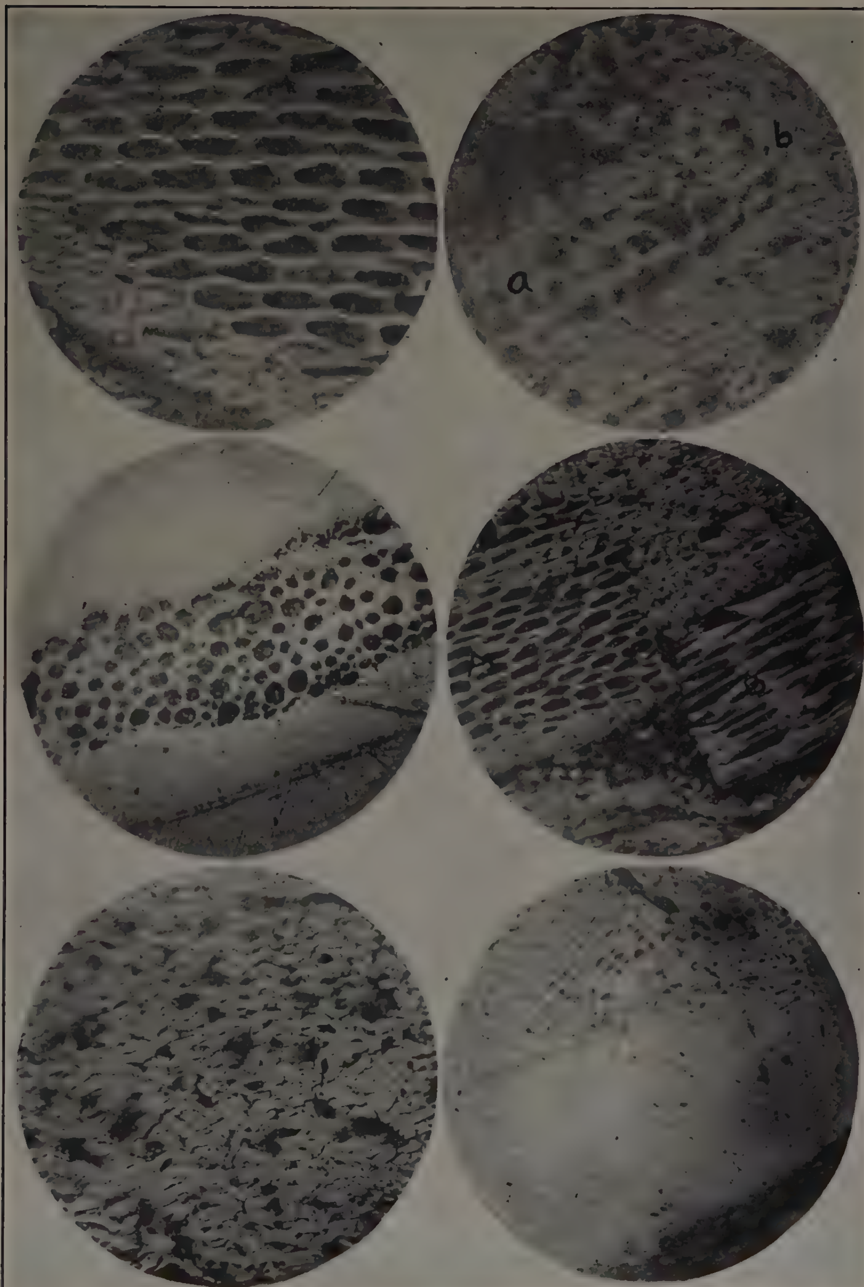
Fig. 24.—Crushed and compressed charcoal. × 150.

Material of this nature is very common in both anthracite and bituminous coal. Intercellular spaces can be seen in some of these fragments.

Fig. 25.—Partly carbonized piece of wood in anthracite. × 150.

Carbonized portion in upper half of figure shows almost perfect cells while in lower portion cells have been flattened. Note that compressed cells are not fractured as in Fig. 19.

20 21
22 23
24 25



FIGS. 20–25.—CAPTIONS ON OPPOSITE PAGE.

The conventionalized drawing in Table 1 is characteristic of both anthracite and bituminous coals, the only difference being the more pronounced conchoidal fracture on the anthracite side of the drawing.

Aside from the charcoal content, the analyses shown here are just what one would anticipate when the constitution of the different laminae is taken into consideration. The brightest layers, which have been shown to be composed of former woody tissue in all cases, should represent, as they do, the portions of greatest purity in both coals. The ash content in these layers probably comes very close to a measure of the inorganic material of the original plant substance.

TABLE 1.—*Chemical Composition of Laminae of Anthracite and Bituminous Coals*

		Anthracite Forge Split Mammoth	Bituminous Freeport bed
Brightest Layers or Anthraxylon	Moist.	1.4	1.7
	V.C.M.[1]	4.8	32.9
	F.C.[2]	92.1	64.2
	Ash	1.7	1.2
	S	0.8	0.6
	Cal.	8233	8094
	B.t.u.	14820	14570
	Sp. Gr.	1.45	1.25
Duller Layers or Attritus	Moist	1.1	0.9
	V.C.M.	5.2	38.8
	F.C.	89.1	53.9
	Ash	4.6	6.4
	S	0.8	0.4
	Cal.	7917	7839
	B.t.u.	14250	14110
	Sp. Gr.	1.48	1.21
Charcoal or Fusain	Moist.	1.2	0.6
	V.C.M.	3.2	23.0
	F.C.	94.3	58.3
	Ash	1.3	18.1
	S	0.4	0.1
	Cal.	8072	5794
	B.t.u.	14530	10430
	Sp. Gr.	1.60	1.73

[1] Volatile combustible matter. [2] Fixed carbon.

The duller layers, being composed of a great variety of plant materials and containing fine mud and other mineral grains should at least show, as they do, a higher ash content than that of the brightest layers.

When we examine the analyses of the charcoal content of both coals, we encounter contrasts which could not have been anticipated from an examination of hand specimens or photomicrographs; for this material has the same macroscopic and microscopic appearance in both anthracite and bituminous coals. One explanation, however, which is compatible with the structural features, is found in the relative lack of cohesion

between the laminae of bituminous coal as compared to anthracite. This permits greater penetration of mineral-charged waters in bituminous coal and hence its charcoal content would have a greater opportunity to absorb inorganic matter. This is further substantiated by the fact that the charcoal of bituminous coal shows a greater content of inorganic carbon dioxide than does the charcoal content of anthracite coal. Of course, if this explanation is true, one is forced to assume that the charcoal of neither anthracite nor bituminous coal was mineralized before the coals had reached the solid stage.

Whereas this may account for the high ash of charcoal from bituminous coal, it does not explain the higher volatile content. One explanation for this condition may be that the charcoal in bituminous coal has taken up gaseous or liquid organic products which have condensed in the pores and have not been released. If we assume that the charcoal in both coals was the same at the outset, it is not difficult to picture them taking up organic gaseous or liquid products at some stage in their metamorphism and releasing them during a later stage. The charcoal of anthracite which has been subject to much greater metamorphism than its bituminous counterpart would thus lose a correspondingly greater percentage of its volatile matter.

One of the most significant things shown in this table is the close relationship between the anthracite components and similar components of bituminous coal. This relationship gives additional support to the theory that anthracite is really bituminous coal which has been subjected to regional metamorphism.

The analyses shown in this table were made by H. M. Cooper, of the Bureau of Mines. They are the results of single analyses of samples prepared in the following manner: Several lumps of coal showing fairly thick bands of anthraxylon were selected from a given bed; a piece of anthraxylon was sawed out of each lump; all these pieces were then pulverized and thoroughly mixed for the analysis. The same procedure was followed for the attritus. The fusain was picked out of that part of the bed that showed a thick layer of this material. The analyses given in Table 1 should, therefore, be fairly representative of the materials of the beds in question.

The gravities given in this table are true specific gravities and were obtained from one specimen only. They may be much farther apart in cases where the duller laminae are much higher in ash than are the brightest laminae.

Separation of Component Parts of Anthracite by Means of Screens

It is reasonable to expect that the three chief laminae of anthracite, with vastly different physical properties as shown herein, would be

separated somewhat if a lump of coal were ground in a mortar and passed through a nest of screens of different mesh. A crude attempt was made to see if this held true. In this test a lump of well laminated anthracite was ground in an iron mortar and shaken in a nest of screens starting with 20 mesh and ending with 100 mesh. The portions remaining on each screen were analyzed with the results as shown in Table 2.

Table 2.—*Component Parts of Anthracite by Screening*

Mesh	Through 20 on 40	Through 40 on 60	Through 60 on 80	Through 80 on 100	Through 100
Moisture	1.4	1.5	1.2	2.2	1.5
Volatile combustible matter	5.0	5.3	4.9	4.1	5.5
Fixed carbon	83.2	82.2	83.1	83.1	79.8
Ash	10.4	11.0	10.8	10.6	13.2
Sulfur	0.5	0.5	0.4	0.5	0.5
Calories	7,233	7,222	7,217	7,217	7,000
B.t.u	13,020	13,000	12,990	12,990	12,600

A few things are suggested by Table 2, although the analyses are so much alike that it may be unsafe to generalize. The coarsest material shows the highest fixed carbon and the lowest ash, suggesting a larger proportion of the brightest laminae, which has been shown to be the purest, most compact, and most homogeneous constituent in anthracite. The finest material shows the lowest fixed carbon and the highest ash which suggests a larger proportion of the duller layers. The charcoal content would probably concentrate in the fines also, but because of its relatively small amount, it would not be reflected in the analyses. Several repetitions of this test on various coals might show more significant results.

Ash and Its Distribution

The mineral content of anthracite and bituminous coals has three sources: namely, (1) inorganic constituents of the coal-forming plants; (2) sediments and dissolved minerals carried into the swamp in which the coal-forming plants grew; and (3) minerals deposited in spaces in the coal after it had reached the solid stage.

The analyses given in Table 1 show that, exclusive of the charcoal, the ash is lowest in the brightest laminae and highest in the duller layers. In other beds where these components could be separated, the ash was found to be as low as 1 per cent. in the brightest bands of anthracite and as low as 0.9 per cent. in the corresponding layers in bituminous coal. Thus the ash percentages in the brightest layers of laminated

bituminous coal or anthracite give a measure of the lowest ash obtainable by mechanical means in a given bed.

It was found, however, that the ash of each lamina of both coals could be greatly reduced by chemical treatment without much alteration of the coal substance itself. Hydrochloric and hydrofluoric acids were tried in various ways until a method was found that gave the lowest ash with the least change in coal substance.

The treatment which gave the best results consisted in boiling the coal for ½ hr. in concentrated hydrochloric acid in an Erlenmeyer flask connected with a Liebig condenser, using 5 g. of 60-mesh coal to 100 c.c. of acid. The sample was then emptied into an 800-c.c. beaker which was filled with cold distilled water, stirred, allowed to stand for 5 min. and then decanted over filter paper into a Buchner funnel in a filtering flask connected to a water suction fitting. The beaker was again filled with distilled water and emptied as before. Five repetitions served to remove all the chlorides as far as could be detected in the wash water. The filter paper was lifted from the funnel and allowed to stand in the room until the coal became dry enough to be brushed into a platinum dish. About 45 c.c. of 49 per cent. hydrofluoric acid was added and the dish was then placed over a low flame to simmer for 2 hr. at a temperature of 95° C., care being exercised to prevent evaporation to dryness. About 30 c.c. of concentrated hydrochloric acid was next added and the boiling continued for 1 hr. The sample was then washed free from chlorides and dried in the manner described above.

TABLE 3.—*Minimum Ash in Coal after Acid Treatment*

	Ash in Untreated Coal, Per Cent.	Ash in Treated Coal, Per Cent.
Brightest band, Freeport	1.20	0.30
Duller band, Freeport	6.40	0.40
Brightest band, Freeport	0.90	0.10
Brightest band, anthracite Mammoth	1.70	0.40
Charcoal, Freeport	18.10	0.90
Anthracite, Primrose bed	13.20	1.10
Anthracite, Skidmore bed	14.77	0.87
Anthraxolite, Sudbury, Canada	30.63	0.41
Anthraxolite, selected piece	0.93	0.09
Rhode Island graphitic coal	19.95	0.46

The minimum ash obtained by this treatment is given in Table 3. The analyses showed a loss of about 1.5 per cent. carbon and 0.1 per cent. hydrogen for anthracite and about 0.6 per cent. carbon and 0.1 per cent. hydrogen for bituminous coal. To go into greater detail concerning this experiment would require more space than the writer

feels is desirable at this time. In a future paper a complete discussion will be given. A similar method was used by A. C. Fieldner and others[5] in the determination of combustible matter in silicate and carbonate rocks.

The results of the acid treatment show that probably almost all of the inorganic matter exists as such and is not combined chemically with the organic radicals.

To obtain further light upon the distribution of the inorganic matter in anthracite, thin sections of it were prepared and boiled in hydrochloric

FIG. 26.—THIN SECTION OF ANTHRACITE WHICH HAS BEEN TREATED WITH ACID AND THEN PARTLY BURNED TO REMOVE SOME OF DARK CARBONACEOUS MATERIAL. × 200.

Dark area with regularly spaced white dots is very bright woody material while rest of area is duller lamina. White dots are spaces formerly occupied by mineral matter. They are slightly enlarged, due to shrinkage during burning. Untreated sections when burned in same way did not show this contrast.

and hydrofluoric acids. It was thought that thin sections of anthracite, which are normally opaque, would thus permit light to pass through the places formerly occupied by mineral matter and therefore show under the microscope just how this inorganic material was distributed.

The treated and untreated sections, however, showed little difference under the microscope. Nevertheless, when these sections were burned, it was very evident that the bulk of the ash material had been removed from the treated section. In the burning it was noticed that the

[5] A. C. Fieldner, W. A. Selvig and G. G. Taylor: The Determination of Combustible Matter in Silicate and Carbonate Rocks. U. S. Bur. Mines *Tech. Paper* 212 (1919) 9.

untreated sections remained compact until they were completely ashed, whereas the treated sections became porous before all the carbonaceous matter had been burned away. On the assumption that these pore spaces represent the locus of places formerly occupied by mineral matter, it was concluded that a photomicrograph would show its distribution. Fig. 26 shows very clearly that the duller lamina contains most of the inorganic matter and also that the smaller content of inorganic matter of the brightest lamina is uniformly distributed.

Adsorptive Properties of Anthracite and Bituminous Coals

A comparison of the nature and properties of anthracite and bituminous coals would hardly be complete without the consideration of their adsorptive properties.

As noted in an earlier paper,[6] bituminous coal from the Freeport bed adsorbed 24 c.c. of CO_2 as compared to 206 c.c. for anthracite from the Forge Split of the Mammoth bed. Again charcoal from the Freeport coal adsorbed 20 c.c. of CO_2 as compared to 159 c.c. for charcoal from Pennsylvania anthracite.

Conclusions

Pennsylvania anthracite and high-rank bituminous coals are almost exactly alike in morphological constitution.

Anthracite is superior to bituminous coal in toughness chiefly because of the elimination of vertical joints through metamorphic reconsolidation.

The three chief components are chemically and physically different in anthracite and in bituminous coal and yet their differences are of the same order when a given component of one is compared to a like component in the other.

The inorganic constituents are not combined chemically with the organic matter to any marked extent.

Anthracite possesses greater adsorptive powers than does bituminous coal.

The fixed ash of both coals is approximately 0.5 per cent.

The bulk of the ash is confined to the duller laminae from which it can be removed in part only after fine grinding, provided a mechanical method is used.

Acknowledgment

The writer wishes to thank A. C. Fieldner and H. M. Cooper, of the U. S. Bureau of Mines, for most of the analyses which appear in this paper; also C. G. Schantz, of Weston Dodson & Co., for many analyses made during this study.

[6] E. Sinkinson and H. G. Turner: Adsorption of Carbon Dioxide by Coal. *Ind. & Eng. Chem.* (1926) **18**, 602.

DISCUSSION

D. WHITE, Washington, D. C. (written discussion).—This paper is interesting not only for the conclusions reached, but also as a demonstration of the value of the Turner process of flame etching of anthracites in the preparation of such coals for paleontological study. Though too far carbonized to show by present methods of section cutting the configuration and detailed structure of the organic debris which, with the associated biochemical decomposition products, makes up the deposit, these coals may by this process be made to reveal more detail than it has hitherto been possible to observe under magnification. It is hoped that the Turner process will be employed on coals by students of the internal structure of fossil plants as well as by those interested in the paleontological details of composition of coals of different ranks. In this connection it may be mentioned in passing that the structure shown in Fig. 21 very likely belongs to some of the fernlike stems abundant in the Paleozoic coal measures.

The demonstrations so admirably presented by Professor Turner corroborate, even in detail, the conclusions that anthracites are the products of further alteration of coals which have progressed through and beyond the bituminous ranks. My conclusions to this effect, reached nearly 30 years ago, were based upon the study of the megastructure of anthracites, to a limited extent, on their microstructure, and on their paleontological associations and depositional relations which are identical with those characterizing the bituminous coals of the same ages, whether Paleozoic, Mesozoic, or Tertiary. All have similar bedding, composition and structure. The tracing of the jetlike lenses and stripes, representing portions of flattened logs, branches and twigs, in coals from the lignite through the subbituminous, bituminous, semi-bituminous and anthracite ranks, has not only been corroborated by Turner, but he makes it possible to study the characters of the detailed cell structure of the different tissues preserved in these brilliant portions of the deposit as they could not be studied before. The deformation and crushing of the tissues in the anthracites as shown by Turner agree exactly with the conditions revealed by Witham, Renault, Thiessen and others, in the woods embedded in the coals of lower ranks.

Though generally masked by crushing or shearing, and possibly by a degree of cementation, mineral charcoal or fusain occurs in anthracites with the same variability, as well as with the same abundance in certain beds, as is found to be the case with coals of lower ranks. My own strong conviction is that, with anthracite as with other coals, much of the dust generated in the mining and preparation of the coal for market is due to trituration of this, the really friable material of the anthracite deposit. I am therefore quite prepared to find that the fixed carbon of the fusain in Dr. Turner's anthracites should be notably larger in amount than that found even in the woody (glance or jetty) layers and lenses, as shown in the analysis by the U. S. Bureau of Mines, quoted by Turner. On the other hand, it remains to be explained why the fixed carbon of the fusain from the bituminous coals is not higher than that of the jetlike wood. Professor Turner's implied suggestion that failure of fusain in the bituminous coal to show the high percentage of fixed carbon which might be expected had it been deposited as charcoal, may be due to impregnation by ulmic and other saturating matters at time of deposition, demands thoughtful consideration. The analysis does not, however, seem to show the relative difference in carbonization which should be reflected by the fixed carbon content if fusain had been laid down as charcoal when the deposit was being formed.

The close examination of Pennsylvania anthracites with the aid of the hand lens reveals the presence not only of carbonized remains, recognizable as belonging to the Lepidophytes, including Stigmaria, and the Calamarian group, but also to the fernlike groups in the fusain layers, unless, so as often happens, due to the friability of these

layers, shearing has taken place along the latter. Also, in rare instances, the grain of the wood forming the jetty layers and lenses may, with favorable illumination, be detected on certain fracture surfaces under low powers. These criteria were included among the evidence on which my conclusions of over 20 years ago were based.

Dr. Turner's photographs furnish additional evidence of the uselessness of further effort to conserve the pseudomineral term vitrain.

Following the presentation of his written discussion Dr. White added that the author's paper is the most complete discussion of the structure of anthracite coal to date, and that the author has evolved a new method of attack and brought it to an excellent stage of development.

A. C. Fieldner, Washington, D. C. (written discussion*).—Professor Turner has presented an important contribution to fundamental knowledge on the constitution and origin of coal. His excellent photomicrographs of anthracite and bituminous coal are convincing evidence of the essential similarity of the morphological constitution of these two ranks of coal. Table 3 shows that the residual inorganic matter left in the coal after extraction with acids is very small. It is usually less than 0.5 per cent. But I question whether it can be concluded that all the extracted inorganic matter existed uncombined chemically with organic matter. I think that some of it—no doubt, a small proportion—may have been combined with organic radicals, as for example, calcium humate. This compound would be decomposed by hydrochloric acid, forming a soluble chloride and insoluble humic acid. It is possible also that small amounts of inorganic matter may be adsorbed in the coal colloid and resist extraction with acids. Without further investigation I would not accept the conclusion that the residual ash of the extracted coal represented inorganic matter combined with organic radicals and that all of the extracted inorganic matter was present in an uncombined state. Further work needs to be done to establish this point.

H. G. Turner stated that there is less than 1 per cent. of mineral matter in the woody tissue and this is either not combined with the coal substance, partly combined or free. The experimental work suggests that it is almost entirely free, but very careful chemical work is required to demonstrate the actual combination of this mineral matter.

A. W. Hesse, Nemacolin, Pa., asked whether seams could be correlated by such examinations of the coal and R. Thiessen, Pittsburgh, Pa., replied that it can be done easily in some cases, but that in others it is quite difficult if not impossible. To this Professor Turner added that an enormous amount of work is required to establish a basis for such correlations, but having established the basis, the matter of correlating should be simple and effective.

---

* Published by permission of the Director, U. S. Bureau of Mines.

Splint Coal*

By Reinhardt Thiessen,† Pittsburgh, Pa.

(New York Meeting, February, 1930)

During the last few years a type of coal called "splint coal" has been brought to the attention of the United States Bureau of Mines, through numerous inquiries concerning its nature. Until recent years it appears to have escaped general notice. The term "splint coal" is of long standing, but was thought to be merely another expression applied by the miner to coal having certain minor characteristics. When closely examined, however, these coals were found to form a marked and distinct type, with distinctive characteristics that cannot be mistaken. Because splint coals produce a hot cheerful flame, possess a clean block-like nature, and have a high volatile content, they have lately become much valued for domestic, gas-producer and steaming purposes, and have attracted considerable attention.

Mattkohle

For many years we have heard from the Germans about *Mattkohle* and *Streifenkohle,* in contradistinction to *Glanzkohle,* but have not realized that they were of a different type from the coals with which we were most familiar. This type of coal was first brought to general notice by Muck, in 1881, who gave a good description of it.[1] Because this coal is often intercalated with thin sheets of brightly shining coal, Schondorff, who was more familiar with the Saar coals, called it *Streifenkohle.* This term was later used by Muck also. Still later Potonié[2] used this term exclusively for the whole group in contradistinction to a class called *Glanzkohle,* and considered these coals to be composed alternately of *Humuskohle* and *Faulkohle* or *Mattkohle* (Fig. 20).

Not knowing the nature of the German coals and not realizing that the term *Streifenkohle* was applied to a distinct class of coal, the terms of its components as used by Potonié and others were translated to "bright coal" and "dull coal" and applied to the components of our

---

* Published by permission of the Director, U. S. Bureau of Mines.

† Research Chemist and Microscopist, Pittsburgh Experiment Station, U. S. Bureau of Mines.

[1] F. Muck: Die Chemie der Steinkohle. Bochum, 1881.

[2] H. Potonié: Die Enstehung der Steinkohle. Berlin, 1910. Bornträger.

coals in general.[3] This was incorrect, and has brought about great confusion in the correlation of terms. The splint coals of America, now known to be of the same type as *Mattkohle*, had not been studied at that time and were not generally known, so that they were not considered in comparison with the German type.

Durain

In England a certain type of coal had always been recognized as "hards" or "dulls." In 1919 Stopes[4] concluded that the different components of coal should be more clearly classified and defined. Four components were then formulated as found in the British bituminous coals—fusain, clarain, vitrain, and durain. It was not realized that durain was a special type of coal.

Durain, from the French *dur*, hard, was a new name assigned to the hards or dulls, and the coal so designated was made the equivalent of the German *Mattkohle* under the following description:

> Durain is hard, with a close, firm texture which appears rather granular even to the naked eye. However straight the break across it, the broken face is never truly smooth, but if looked at closely, always has a finely lumpy or matte surface. Generally, even in the dullest of durain bands, a few (or many) flecks or hair-like streaks of bright coal are to be seen.

When in England in 1925 and 1926, the writer had an opportunity to make a microscopic examination of the durain of the British coals. He found that durain is a characteristic component of coal in which the attritus is composed essentially of an opaque constituent; in other words, an opaque constituent is the characterizing ingredient to which durain owes its characteristic nature.

A later investigation of the German *Mattkohle* or *Streifenkohle* showed that it also owed its specific and characteristic nature to a similar opaque matter. Now it is found that the splint coals of America are of the same type.

Similarity of Splint Coal, Durain, and Mattkohle

We have, then, a type of coal which in America is called splint coal; in England, hards or dulls, now also durain; in the Ruhr district of Ger-

---

[3] R. Thiessen: Compilation and Composition of Bituminous Coals. *Jnl. Geol.* (1920) **28,** 185–209.

Structure in Paleozoic Bituminous Coals. U. S. Bur. Mines *Bull.* 117 (1920) 296.

[4] M. C. Stopes: On the Four Visible Ingredients in Banded Bituminous Coal. *Proc.* Roy. Soc. (1919) **B90,** 470–487.

many, *Streifenkohle* and *Mattkohle,* now also "durit;" and in the Saar district, *Streifenkohle* and *Mattkohle* or *houille matte* in French. Undoubtedly the same types of coals are found elsewhere. These coals are so similar that a description of the one will describe the other, except perhaps in a few minor details.

For the study of the British coals, those of the Beeston bed were taken as a basis. An examination of this coal was made from top to bottom of the vein. The bed has a number of typical durain bands distributed through it which are sometimes as much as 2 to 3 in. thick. Coals from the Grassmore colliery, the Hatfield and East Kerkby collieries in the Barnsley bed, the Plessy bed, the Low Main bed, the Parksgate bed, and the Wigan Four-Foot bed, and a number of other samples of unknown or uncertain origin, were studied for comparison.

The study of the Ruhr coals was made chiefly on samples sent by Dr. Kukuk of Bochum, Dr. Broche of Essen, and Dr. Fischer of Mühlheim from Flöz Zollverein, König Ludwig mine, Flöz Bismark or No. 15, Baldur mine, and Flöz Aegis, Baldur mine. Rittmeister reported that every bed examined in the Ruhr Basin contained *Mattkohle* in bands of varying thickness and in amounts of 10 to 50 per cent. of the coals.[5] In the Saar coals the *Mattkohle* also occurs in bands varying in thicknesses up to 3 cm. It is most common in the Clarenthal coals,[6] where it constitutes the larger part of the beds.

The study of the American splint coal was carried out mainly on samples from the Highsplint bed at Closplint, Ky. Samples from the Kellioka bed, Benham, and the Harlan bed, Chevrolet, Harlan County, Ky., and from the Hernshaw bed, Boone County, W. Va., as well as other samples of doubtful origin, were examined for comparison.

Splint coal is also reported to be found in the following beds: Flag, Leonard, No. 2 Gas, and Thacker, in Kentucky; Millers Creek, Dorothy, Coalburgh, Island Creek, No. 2 Gas, Stockton, Lewiston, Thacker, and Winifred, in West Virginia.[7]

Composition of Coal in General

Before it is possible to comprehend the underlying principles upon which the grouping or classification of these coals is based it will be necessary to give a brief review of the structure of coal in general. To do

---

[5] W. Rittmeister: Eigenschaften und Gefügebestandteile der Ruhrkohlen. *Glückauf* (1928) **64,** 589–594, 624–637.

[6] H. Hoffmann: Die makroskopischen Gemengteile der Saarkokskohle. *Glückauf* (1928) **64,** 1237–1243, 1273–80.

[7] Keystone Coal Buyers Catalog, 1929. New York, McGraw-Hill and Directory Co., Inc.

this it will be necessary to repeat from articles and papers previously given by the writer.[8]

All coals are essentially composed of two visibly different classes of constituents—anthraxylon and attritus (Fig. 1).

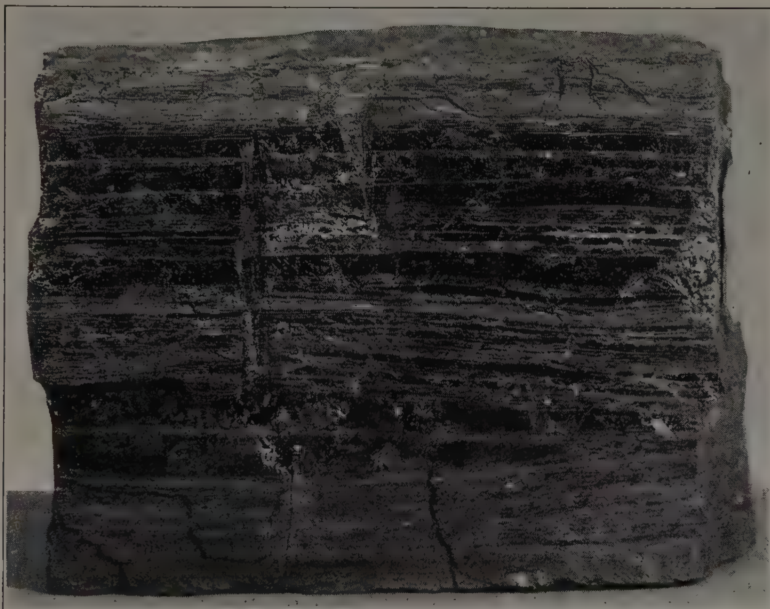


Fig. 1.—Block of banded coal from Upper Freeport bed, Freeport, Pa. Two-thirds natural size.
Black bands are anthraxylon; the gray are attritus.

Anthraxylon

Anthraxylon comprises those constituents in coal which are derived from the woody tissues of plants, such as of stems, limbs, branches,

[8] R. Thiessen: Constitution of Coal Through a Microscope. *Proc.* Coal Min. Inst. of Amer. (1919) 34–45.

Compilation and Composition of Bituminous Coals. *Jnl. Geol.* (1920) **28,** 185–209.

Recent Developments in Microscopy of Coal. *Coal Age* (1920) **18,** 1183–1189, 1223–1228, 1275–1279; (1921) **19, 12**–15.

Recent Developments in the Microscopic Study of Coal. *Proc.* Coal Min. Inst. of Amer. (1920) 88–121.

Structure in Paleozoic Bituminous Coals. U. S. Bur. Mines *Bull.* 117 (1920) 296.

Origin and Constitution of Coal. *Proc.* Wyoming Hist. and Geol. Soc., Wilkes-Barre, Pa., 1924.

The Microscopic Constitution of Coal. *Trans.* A. I. M. E. (1925) **71,** 35–116.

The Microstructure of Coal. *Jnl.* Roy. Soc. Arts, London. (1926) **74,** 535–557.

Also D. White and R. Thiessen: The Origin of Coal. U. S. Bur. Mines *Bull.* 38 (1913).

twigs, and roots, including both wood and cortex, changed through decay and coalification processes, but still present as definite unit constituents. With the naked eye they appear as homogeneous black

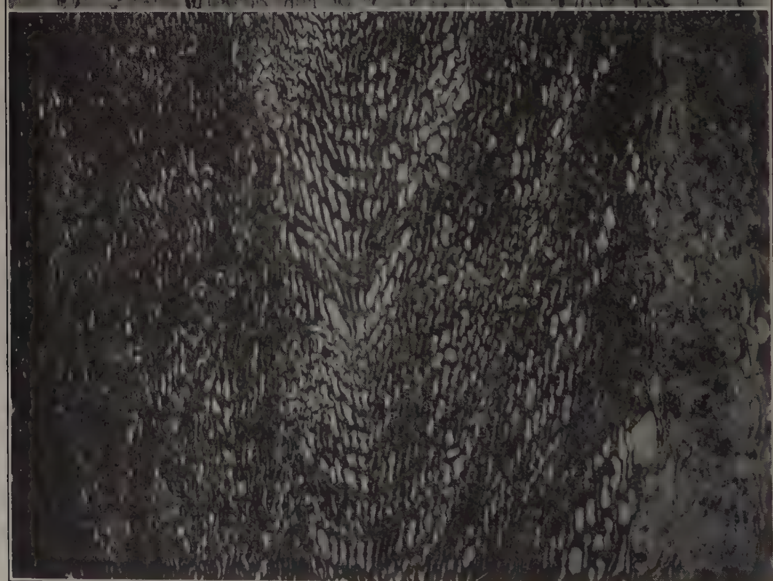


Fig. 2.

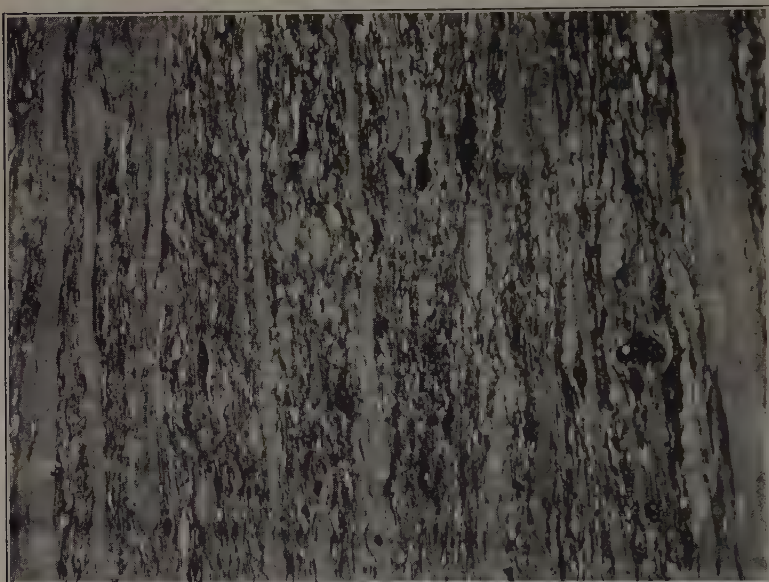


Fig. 3.

Fig. 2.—Thin cross-section of anthraxylon of a Pittsburgh coal. Wood cells filled with resinous matter. × 200.

Fig. 3.—Thin cross-section of Pittsburgh coal showing typical attritus and thin strips of anthraxylon. × 200.

Attritus is composed largely of humic degradation matter, many spores, some resinous matter and opaque matter, and a few fragments of fusain.

Reduced one-third; original magnifications given.

bands, strips or lenticular inclusions, usually of a bright outward appearance but sometimes varying from a smooth black to a highly lustrous black, according to the stage of metamorphism of the coal; the more

highly metamorphosed the coal, the higher the luster. Sometimes the anthraxylon may be quite devoid of luster.

In most respects the anthraxylon corresponds to the British vitrain, and in part to the German *Glanzkohle*.

Microscopic examination of thin sections of these bands always reveals some of the original cell structure, which appears very faint in some sections and well preserved in others (Fig. 2), with all gradations between.

Attritus

Attritus is that component in coal derived from any or all plant matter which contributed to the deposit in the peat stage, macerated and comminuted through the agency of micro-organisms, lower forms of animal life, and the meteorological influences, and subsequently changed into coal; therefore attritus consists chiefly of the more and most resistant plant products.

Attritus is of much duller appearance than anthraxylon, being usually of a grayish dull color, often intercalated with fine hairlike streaks of bright coal (Fig. 1). A typical attritus of an ordinary coal is shown in Fig. 3.

The attritus in coal is analogous to the black muck in peat. Peat muck is really an attritus and is composed of the same kind of constituents as coal attritus. Knowledge concerning it is therefore valuable in understanding coal, and makes possible a precise interpretation of the constituents in coal.

Peat Attritus

Peat attritus is the general residue left from all possible kinds of plants, plant products, and plant organs through the agency of fungi, bacteria, actinomyces, lower forms of animal life such as insects and their larvae, crustacea and worms, and meteorological influences, all active on the contributing plant matter largely while it is freely exposed to air. Following are the chief constituents:

1. *Humic Degradation Matter.* (*a*) *Woody Degradation Matter.*—When wood is left to decay in moist air, much of it is in time reduced to a more or less finely macerated product. The cellulose of tissues is readily decomposed into water, gases and soluble acids, while the lignin, in some manner changed into substances called humus or humic acids, largely remains. The wood has been reduced to a granular mass in which may be recognized bits of tissues and fragments of wood cells or fibers.

(*b*) *Degradation Matter from Other Carbohydrate Tissues.*—Another type of degradation matter is derived from leaf tissues, petioles, cortex, bark, pith, etc., and from the tissues of such plants as ferns, mosses,

liverworts, lichens and fungi, macerated through agencies of micro-organisms, lower forms of animal life, and weathering. Again, as in the woody tissues, the cellulose of all tissues has been largely decomposed and has disappeared, while the lignin transformed into humus has largely remained.

In peat the woody degradation matter and that of the other tissues are definitely distinguishable under the microscope, yet, as they are similar in their chemical behavior, in the further transformation from peat into successively higher grades of coal they become similar to each other in appearance and distinguishable with difficulty. These two derivatives are therefore together classed as humic degradation matter and with the larger pieces of woody peat largely constitute the humus and humic acids of peat.

2. *Spores and Pollens.*—Spores are asexual reproductive organs of pteritophytes, mosses, liverworts, and fungi. The spores of pteritophytes are chiefly important in coals. In the living plant the spores are more or less spherical. They consist of an inner living part, the protoplasm and its inclusions; this is surrounded by a thin membrane, the entine, which in turn is enclosed by a relatively thick outer coating, the exine. The exine is very resistant to all chemical reagents, micro-organisms, and to weathering, and alone resists the peat-forming and coal-forming agencies, while the inner cell contents and the entine disappear.

Pollens are similar to spores in function, structure and chemical behavior; the outer wall or exine is much thinner than that of the spores, although it also resists the peat-forming and coal-forming agencies.

3. *Cuticles.*—All leaves, petioles and growing parts of the higher plants are covered by a protective membrane to guard them from the attacks of micro-organisms, insects and the weather. As in the spore and pollen exines, cuticles are very resistant to chemical reagents and micro-organisms, and therefore survive in a more or less fragmented state and are constantly present.

4. *Resins.*—All plants contain some resinous matter either in the cell or in fissures or cavities of the tissues of leaves, wood and bark of the plants. Living conifers now contain the greatest quantity of resins. These are not decomposed by micro-organisms and therefore survive in peat and throughout the following ranks of coal, either as free particles or lumps or as inclusions in the remaining wood, bark and leaf tissues.

5. *Waxy Matter.*—Many plants contain waxes, either in certain cells of the tissues as films, or as layers in the form of granules, heirs, or rodlets on the surface of leaves and green stems. Such waxes are also resistant to chemical reagents, micro-organisms and weathering, and constantly remain in peat.

6. *Opaque Matter.*—Careful microscopic examinations of a peat attritus will reveal a certain ingredient which in comparison with the

other constituents is darker in appearance and not nearly so transparent, and which in thicker layers or larger fragments is totally opaque. Its origin has not been definitely determined and for want of a better name it is provisionally termed "opaque matter."

These various constituents are always present in an ordinary peat attritus, but not always in the same proportions. As a rule, the humic degradation products furnish by far the largest part of the attritus, but in certain types of peat other constituents may form important parts.

Same Constituents Found in Coal

All of these constituents can be followed step by step from peat through all the ranks of coals—brown coals, lignites, bituminous coals, and anthracite—so that their identity and origin in the coals of higher rank is well established. In all coals, therefore, the major constituents that can be recognized definitely are anthraxylon and attritus. The attritus again is composed of (1) humic degradation matter, (2) spores and pollens, (3) cuticles, (4) resins, (5) waxes, (6) opaque matter, (7) mineral matter; and to these should be added (8) oil algae and (9) fusains or mineral charcoal.

Although, genetically, some fusain should be classed with the anthraxylon, some of it is plainly of other origin. Because of its specific difference in physical structure, fusain is customarily put in a class by itself. To these definitely recognizable visible constituents must be added a number of other constituents invisible to the microscope yet present in an absorbed or finely dispersed state, such as pigments and other glucosides, alkaloids, terpenes, etc.

Compilation

The relative proportions of the two major components—anthraxylon and attritus—play an important part in the nature of a coal. Ordinarily, coals are composed of a mixture in all possible proportions. Common appearances of anthraxylon and attritus under the microscope at a medium high magnification are shown in Fig. 4. Coals in which the relative amount of anthraxylon is high, are termed "anthraxylous" coal, as that in the lower bench of the Upper Freeport; other coals in which the relative amount of attritus is high, as in the Pittsburgh coal, are termed "attrital" coal. When a coal contains none or only relatively little anthraxylon, like that in the Kittanning bed, it is a cannel coal; in other words, a coal composed entirely or almost entirely of attrital matter and assuming specific characteristics, is called a cannel coal, as found in the Kittanning bed. The nature of a cannel coal is determined by the predominance of the one or the other of its constituents.

While it is true that the relative amounts of anthraxylon and attritus determine the nature of the coal, the predominance of the one or the

other of the constituents composing the attritus also determines the nature of the coal.

FIG. 4. FIG. 5.

FIG. 4.—THIN CROSS-SECTION OF UPPER FREEPORT COAL, SHOWING ANTHRAXYLON BANDS ALTERNATING WITH LAYERS OF TYPICAL ATTRITUS. × 200.

Attritus is composed of spores and humic degradation matter and some opaque matter.

FIG. 5.—THIN CROSS-SECTION OF PITTSBURGH COAL IN WHICH ATTRITUS IS COMPOSED LARGELY OF OPAQUE MATTER. × 200.

Notice that opaque matter or matrix appears as a homogeneous mass in which spores appear to be embedded. Gray strips are anthraxylon and humic matter.

Reduced one-third; original magnifications given.

These eight or nine constituents of the attritus named may be present in all imaginable proportions. All may be present in about equal amounts, or one or the other may predominate, or two or three in about equal proportions may form the larger part of the attritus—each arrange-

ment lending a specific characteristic to the coal as a whole. One or several of the components may be absent but never is one present alone.

Possible Composition of Attritus

Although in many coals the various components are present in more or less equal proportions, there are a number of coals where one or the other does predominate and to which the coal owes certain specific characteristics; for example:

1. Attritus of a coal may be composed largely of humic degradation matter. A notable example of such a coal occurs in the Redstone bed of the Monongahela series, in which the attritus is composed largely of humic matter and contains relatively few spores, resinous particles, cuticles or opaque matter.

2. Attritus may be largely composed of spores. A conspicuous example of such a coal is found in certain layers of the Pittsburgh bed, which, as a whole, is rich in spore matter. The organic matter of spore-cannel coals also is largely composed of spore matter.

3. Attritus may contain a large proportion of cuticles. Not many coals of this type are known, but the Russian coal called *Papier Kohle*, of *Malowka*, which, according to Zeiller,[9] consists of an accumulation of the cuticles of Bothrodendron, is a good example. Another example is found in the Dusonich de Lievin bed of France.[10]

Many coals have thin seams containing numerous cuticles, as in an Illinois coal.

4. The attritus may contain a large proportion of resins. Coals of this type more often occur in the younger horizons. A coal from Sunnyside, Utah, is a good example.

5. No ordinary coal has as yet come to our knowledge in which attritus is composed chiefly of waxes, but certain oil shales like that of Soldiers' Summit, Utah, come under this classification.

6. Oil algae have not yet been found in the ordinary American coals. In certain British coals they are sparingly represented, but the boghead[11] coals are largely composed of oil algae.

7. The opaque matter is the most important constituent for the present discussion, as it is the constituent to which this group of coals owe their characteristic nature.

---

[9] M. R. Zeiller: Observations sur Quelques Cuticules Fossiles. *Ann. des Sci. Nat.* [6 Bot.] (1882) **13,** 217–238.

[10] A. Duparque: Les Charbons de Cuticles du Basin Houiller du Nord de la France. *Ann.* Soc. Geol. du Nord (1927) **52,** 2–27.

See also V. H. Legg and R. V. Wheeler: Plant Cuticles, II.—Fossil Plant Cuticles. *Jnl.* Chem. Soc., London (1929) 2449–2458.

[11] R. Thiessen: Origin of Boghead Coals. U. S. Geol. Survey *Prof. Paper* 132-I, in Shorter Contributions to General Geology (1925) 121–128.

Classes of Opaque Matter

In general, two classes of opaque matter may be recognized: One class has definite structure and clearly represents highly carbonized parts of cells or small groups of cells and bits of tissues. This class is of the same general nature as fusain and may be placed under the same head. The identity of this class is easily established through the various

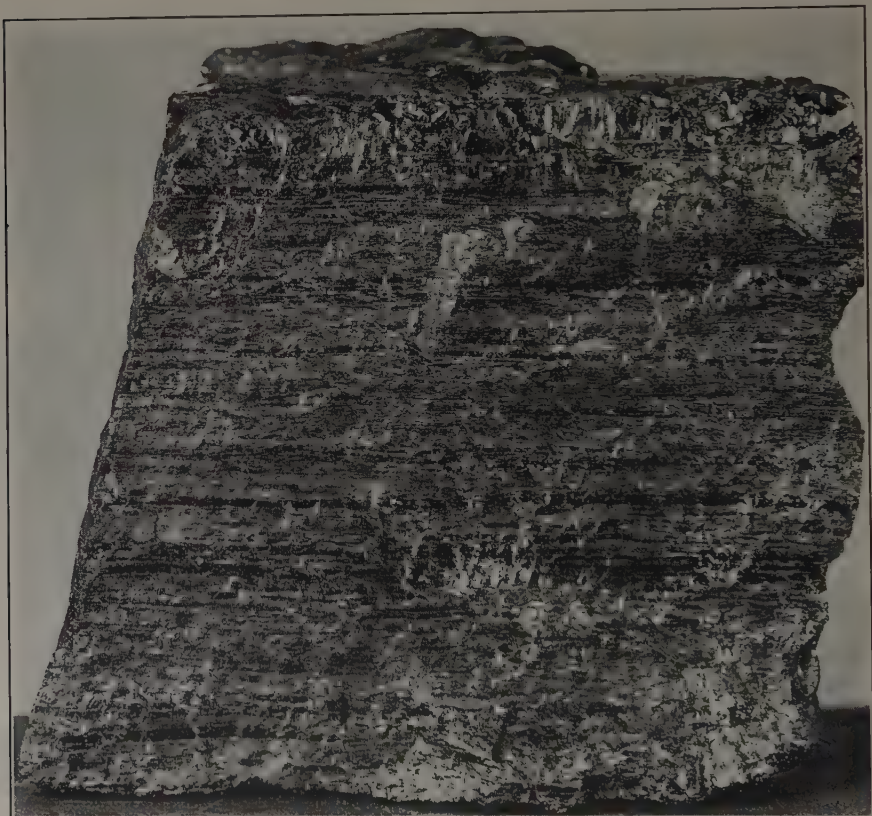


FIG. 6.—BLOCK OF SPLINT COAL FROM HIGHSPLINT BED, CLOSPLINT, HARLAN CO., KY. TWO-THIRDS NATURAL SIZE.
Notice thin sheets of anthraxylon in black, distributed through block.

cell characteristics. The other class consists of irregular and disorganized plant matter, in which little or no cell structure is evident. This class is not so easily defined as the former and is apparently of special origin. It occurs in a large variety of forms and shapes of particles. They vary from the tiniest microscopic particle, colloidal in size, to such as are visible to the unaided eye. In a medium-thin section they are opaque. In a very thin section many are translucent or even transparent, and present a dark ash-gray red color by transmitted light, but many remain

opaque. Taken as a whole, these particles, which collectively constitute an opaque matrix, vary considerably in their opacity. In the same section, side by side, may be found particles of all degrees of opacity, ranging from those that are but slightly more opaque than the particles composing ordinary humic attritus, to those only translucent in the thinnest section

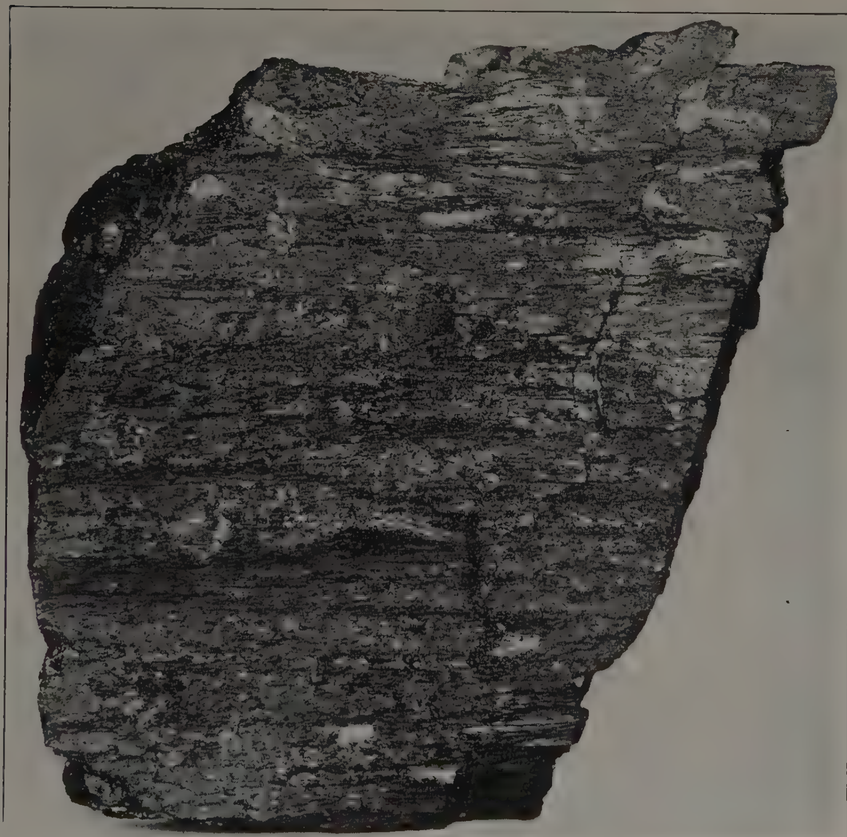


Fig. 7.—Block of Mattkohle (durit) from Zollverein bed, König Ludwig mine, Ruhr, Germany. Two-thirds natural size.
Notice anthraxylon (Glanzkohle) scattered as very thin strips throughout block.

possible or totally opaque. On the whole, the higher the rank of coal, the less translucent are these particles.

The opaque matter is present in all coals in greatly varying proportions; in some there may be relatively little dispersed throughout the coal, in others there may be considerable amounts in certain thin layers, as in some layers of the Pittsburgh coal, but never enough to give the coal the character of a splint coal (Fig. 5). In some coals the opaque matter is predominant in certain thicker layers; in others the attritus of the whole bed is predominantly composed of opaque matter. To

such coal beds, or to such layers of a bed, the opaque matter has given definite and specific characteristics, and as long as coals have been used they have been designated by specific terms.

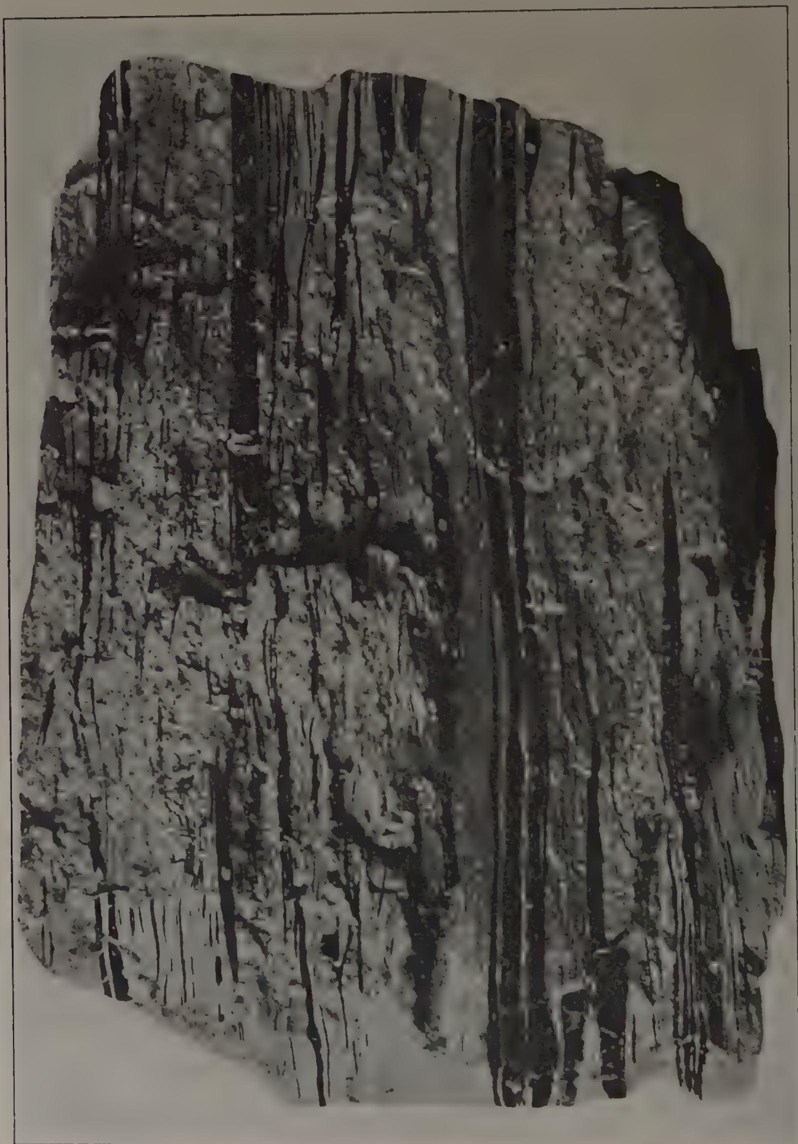


Fig. 8.—Block of Mattkohle, or Streifenkohle, from Saar district, St. Ingbert mine, Phalz. Natural size.
Black strips are Glanzkohle (anthraxylon); lighter parts are Mattkohle (attritus); *a* is Faserkohle (fusain). (Fig. 29 of H. Potonié: Die Enstehung der Steinkohle.)

These are the coals already referred to in the United States as splint coals (Fig. 6); in the Ruhr district of Germany (Fig. 7) and in the Saar coal fields of France as *Mattkohle*, after Muck, and *Streinfenkohle* after

Schondorff, now durit (Fig. 8); and in England as hards and dulls and recently as durain, after Stopes.

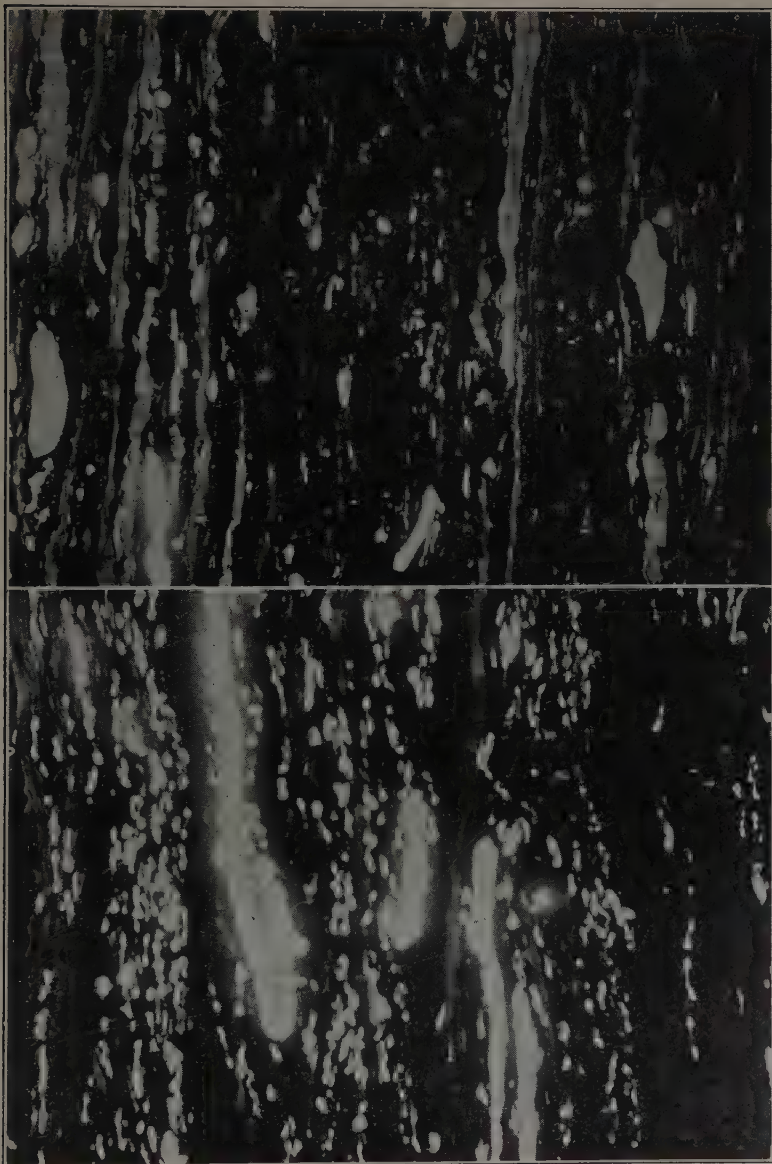


Fig. 9. Fig. 10.

Fig. 9.—Thin cross-section of Mattkohle, Flöz Zollverein, König Ludwig mine, Ruhr, Germany. × 200. Attritus is composed of opaque matter in which are embedded numerous microspores and several megaspores.

Fig. 10.—Thin cross-section of Splint coal, Highsplint bed, Closplint, Harlan Co., Ky. × 200. Shows general appearance of a thin section of attritus. Attritus includes a few thin strips of anthraxylon and fragments of humic matter, and is composed mainly of opaque matter and some spores. Notice that opaque matter appears as a homogeneous mass in this section.

Reduced one-third; original magnification given.

Structure and Composition of Splint Coals

Splint coals, hards or durain, and *Mattkohle* or durit differ radically from the ordinary humic coals in their dull gray color, granular consist-

ency, great hardness, solidity, toughness, higher specific weight, chemical composition and behavior on chemical treatment. Since the term

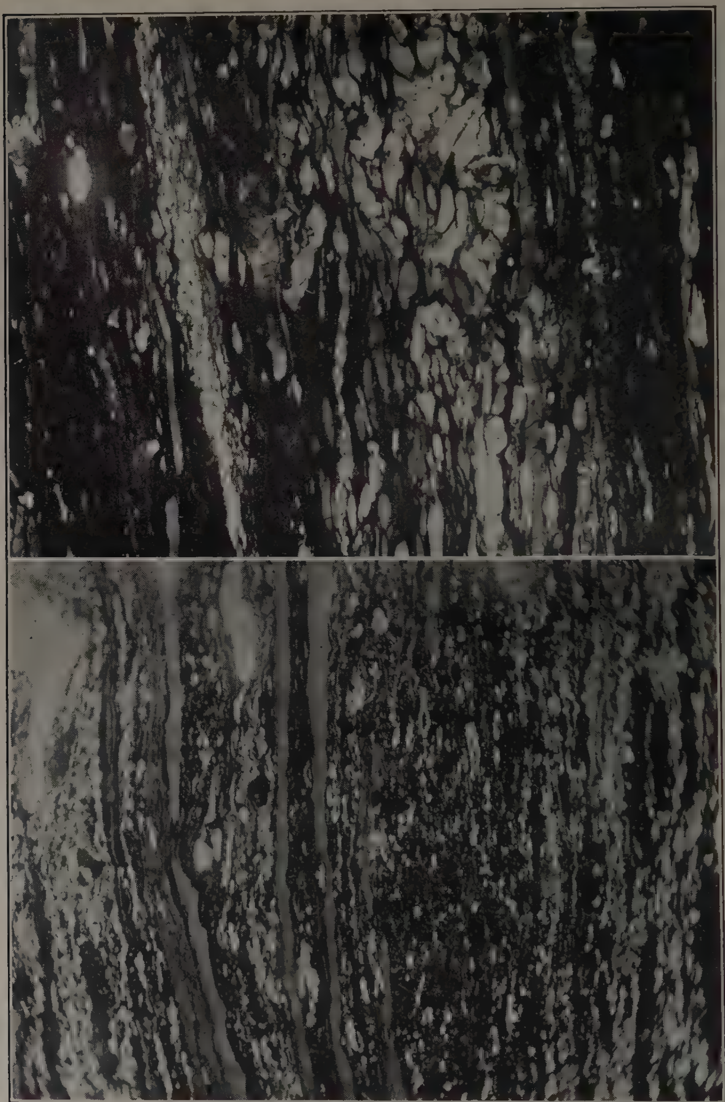


Fig. 11. Fig. 12.

Fig. 11.—Thin section of durain from Beeston bed, Yorkshire, England. $\times$ 200. Shows thin strips of anthraxylon (vitrain) embedded in attritus. Attritus is chiefly composed of opaque matter in which are embedded numerous spores.

Fig. 12.—Thin cross-section of splint coal from the Highsplint bed. $\times$ 200. Contains a larger proportion than Fig. 10 of humic matter embedded in opaque matter. Reduced one-third; original magnifications given.

"splint coal"[12] has been thoroughly incorporated in usage and the literature, and is quite well understood, "splint coal" will be used in this paper to designate the whole group of coals coming under this type.

[12] The Century Dictionary defines splint coal as a variety of bituminous coal which is of a dull, stony luster and breaks in slablike masses.

When a piece of typical splint coal (durain, or *Mattkohle*) is ground on a glass plate with a fine abrasive, the ground surface assumes a brown-

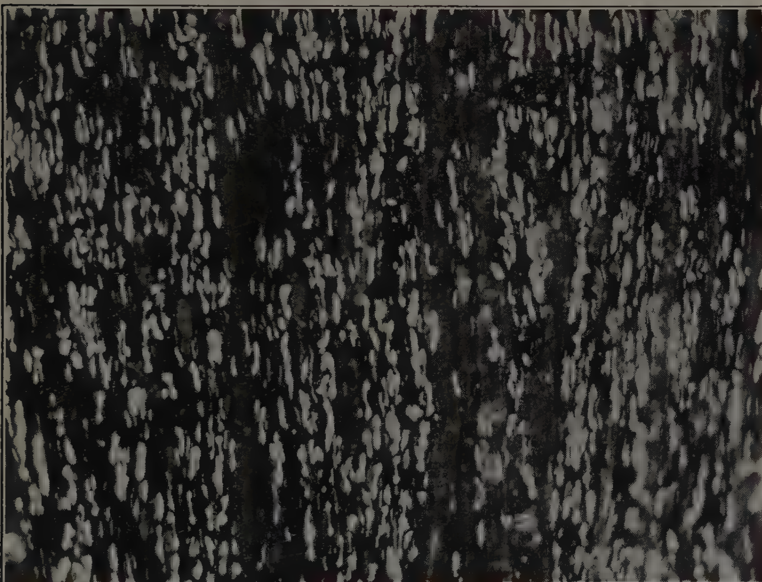


Fig. 13. Fig. 14.

Fig. 13.—Thin cross-section of Mattkohle (durit) from Flöz Zollverein. × 200. Still larger number of anthraxylon strips than in Fig. 12.

Fig. 14.—Thin cross-section of Lower Kittanning coal. × 200. Shows thin anthraxylon bands embedded in opaque matrix, containing numerous spores. Notice form of spores. According to composition and structure this should be classed as a splint coal.

Reduced one-third; original magnifications given.

ish black appearance; when an attempt is made to polish it to a high degree, it assumes a dull grayish black appearance, but never acquires the high polish assumed by ordinary coals treated in the same manner.

Appearance of Thin Section

Sections thin enough to be transparent are difficult to prepare from these coals. In a section in which the attritus of an ordinary coal is transparent splint coals show a dark opaque substance in which a number of spores are embedded and in which possibly fragments of cuticles

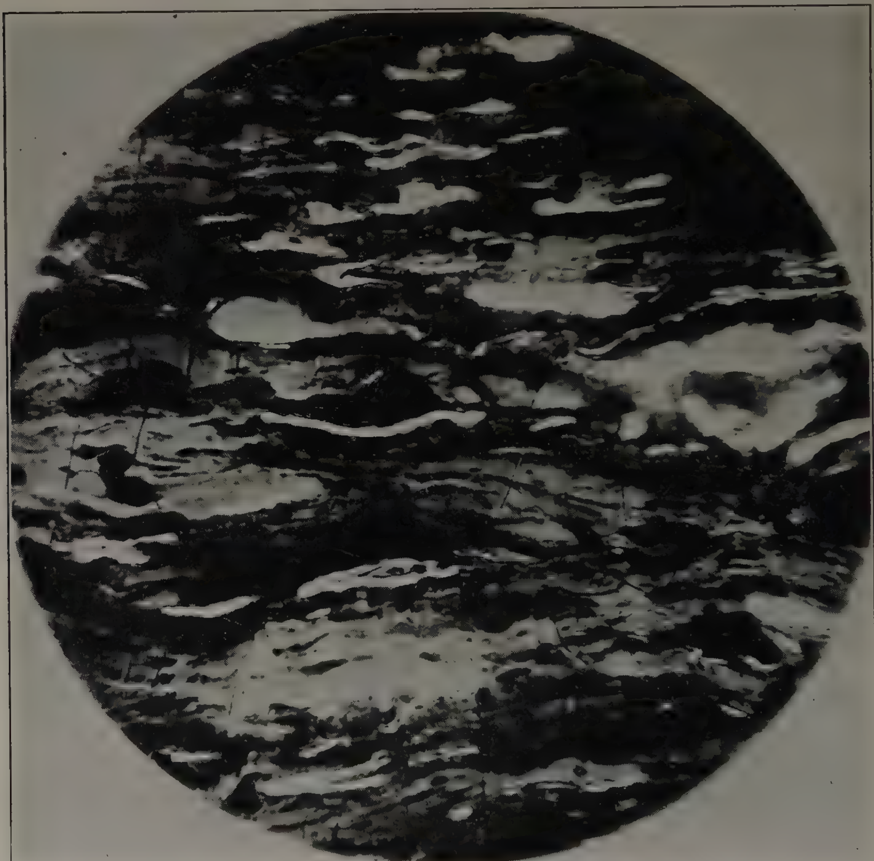


FIG. 15.—THIN CROSS-SECTION OF SPLINT COAL, HIGHSPLINT BED, CLOSPLINT, KY. × 1000.
This is a thin section and opaque matter has resolved into a heterogeneous mass; it is shown to consist of more or less definite particles of greatly varying transparency.

stand out here and there in great contrast to the black background. Next, as a rule, a varying number of dark red humic particles and resinous globules are seen. A few thin strips of dark red anthraxylon may also be observed. This is the most common picture, of which Figs. 9, 10 and 11 show typical examples. As the section is moved along, or in other sections, the appearance may vary. There may be a larger number

of humic particles (Fig. 12); or a larger number of anthraxylous strands, as in Fig. 13; or there may be a larger or a smaller number of spores, as in Figs. 9, 10, 13 and 14.

In some parts of a section the humic particles may predominate (Fig. 12), but this is never the rule. Whatever the amount of anthraxylon or whatever the amount of humic particles, they are always of a deep red color and the attritus is always relatively opaque.

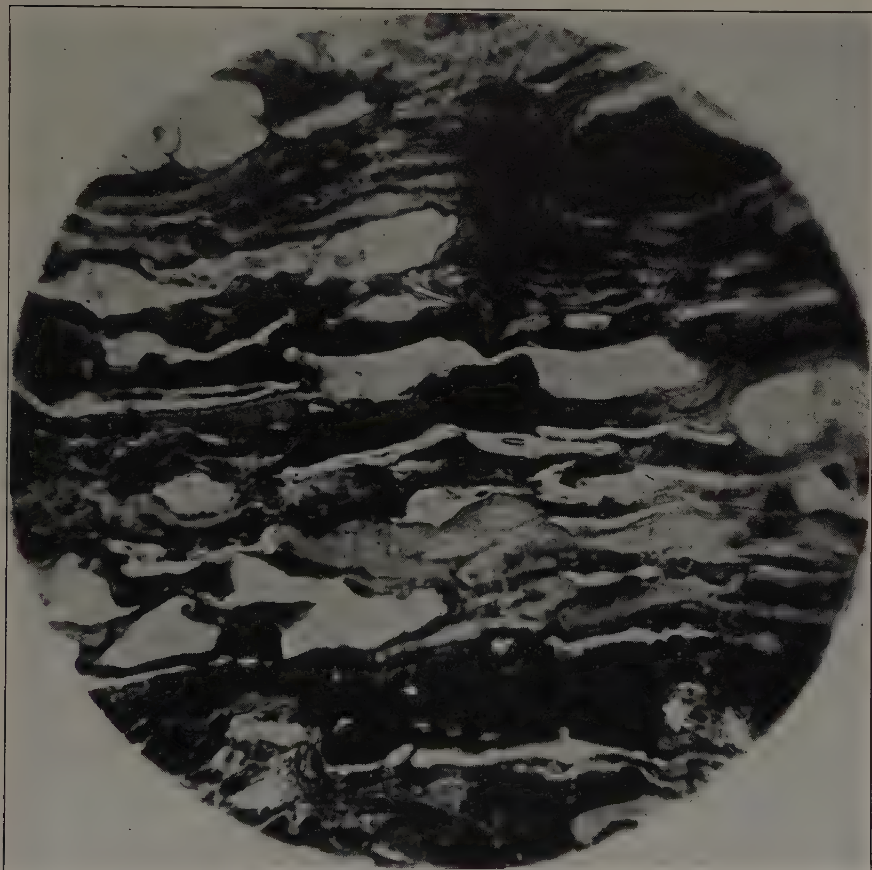


FIG. 16.—THIN CROSS-SECTION OF MATTKOHLE (DURIT) FROM FLÖZ ZOLLVEREIN, KÖNIG LUDWIG MINE, RUHR, GERMANY. × 1000.
From very thin section showing nature of opaque matrix. Notice form of spores. Reduced one-third; original magnification given.

As the sections are ground thinner, more and more of the constituents of the attritus become translucent or transparent, and when made very thin, the opaque matter is revealed as a heterogeneous mass. When finally the section is made exceedingly thin, most of the constituents

become transparent or translucent; they then appear as a heterogeneous mass under the microscope, but some remain opaque, no matter how thin the section may be. When examined at a high magnification, the constituents consist of more or less definite unit particles varying greatly in form, size and transparency (Figs. 15, 16, 17 and 18).

FIG. 17.—THIN CROSS-SECTION OF MATTKOHLE FROM FLÖZ ZOLLVEREIN, RUHR DISTRICT. × 1000.
Shows type of spores predominating in this type of coal.

Anthraxylon

As already shown in the block and thin section, the splint coals usually contain numerous thin bands or sheets of anthraxylon, a characteristic that merited them the name *Streifenkohle*. Relatively few bands are found more than a few millimeters thick, so that the total amount of anthraxylon is much less than in ordinary coals.

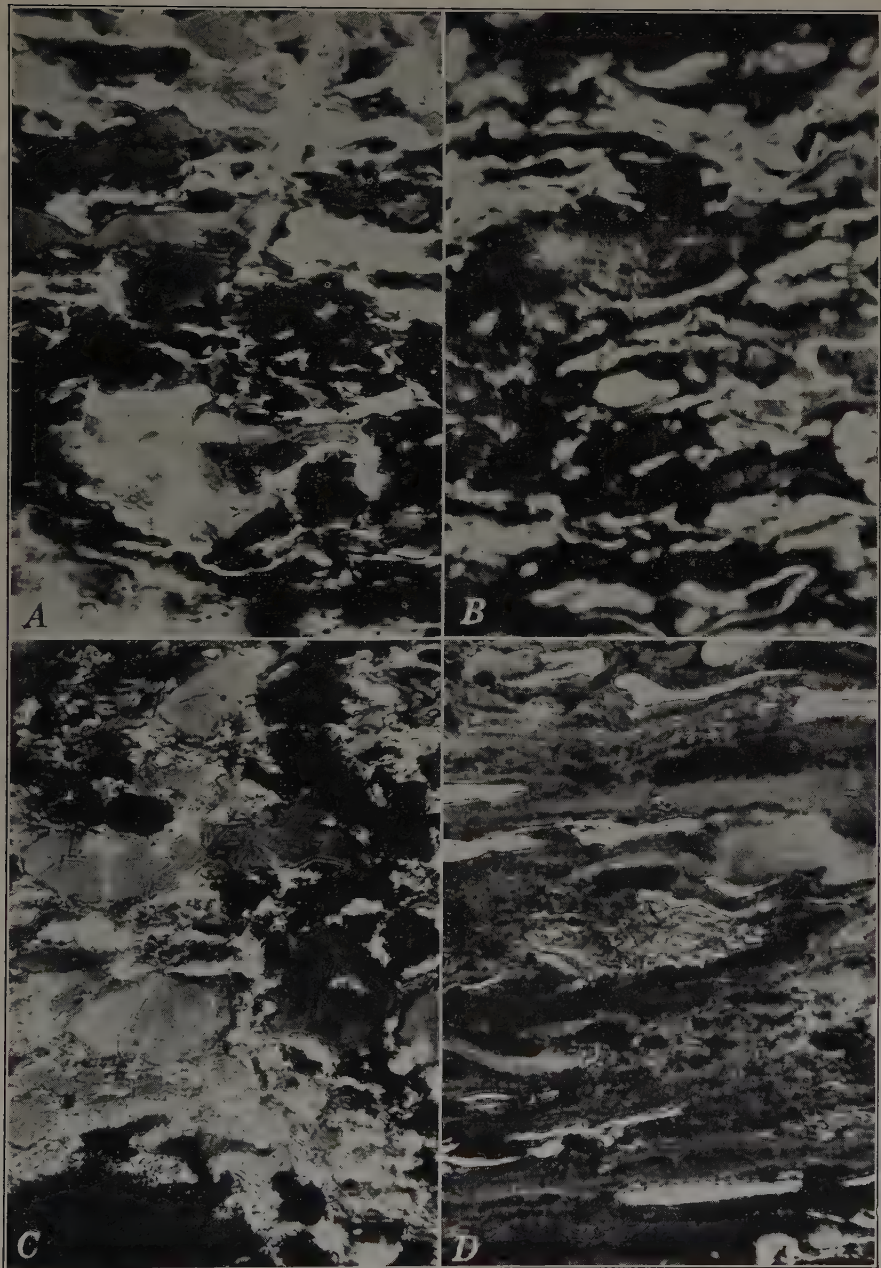


FIG. 18.—THIN CROSS-SECTIONS OF ENGLISH DURAINS. × 1000.

A from Barnsley bed, Kirkby mine
B from Plessy bed
C from Barnsley bed, Hatfield mine
D from Wigan 4-ft. bed

Reduced one-third; original magnification given.

The distribution, number and thickness of the anthraxylon sheets are well shown in the illustrations of the blocks. Some of these are thinner than sheets of paper; many are of microscopic thickness and

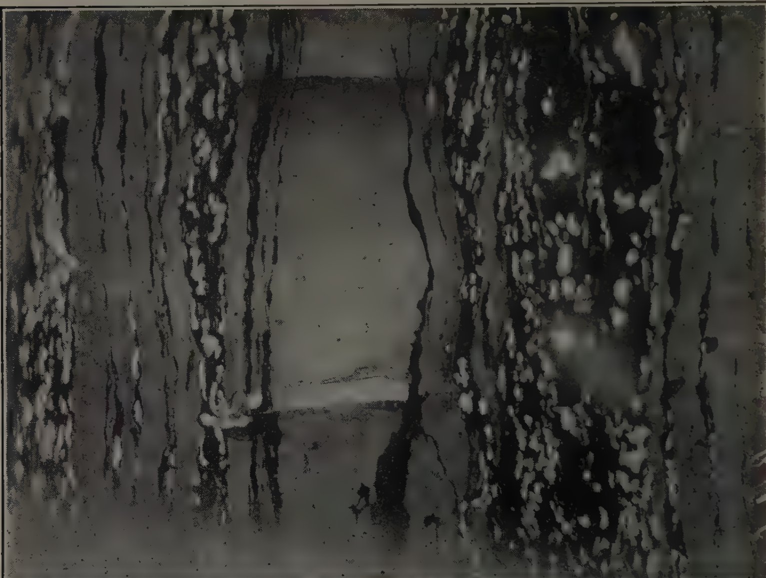


Fig. 20.

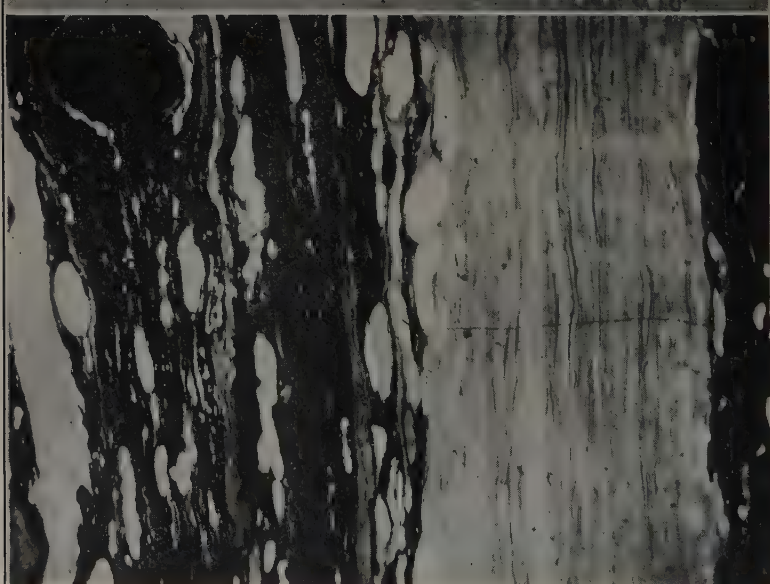


Fig. 19.

Fig. 19.—Thin cross-section of splint coal, Highsplint bed, Closplint, Ky. × 200. Shows anthraxylon band in which woody structure is shown as horizontal parallel striae. Larger oval white spots are resin particles.

Fig. 20.—Thin cross-section of Mattkohle (durit) from Flöz Zollverein, Baldur mine, Ruhr, Germany. × 200. Shows anthraxylon with well-preserved cell structure.

Reduced one-third; original magnifications given.

are only seen in thin sections under the microscope. Many of them are thin lenticular sheets. A peculiarity of the anthraxylon strips is that they are irregular in shape, with crooked edges (rarely straight), and

often are divided or frayed out on the ends (Figs. 11 and 13). They are always deep red in color, becoming lighter in thinner sections; as

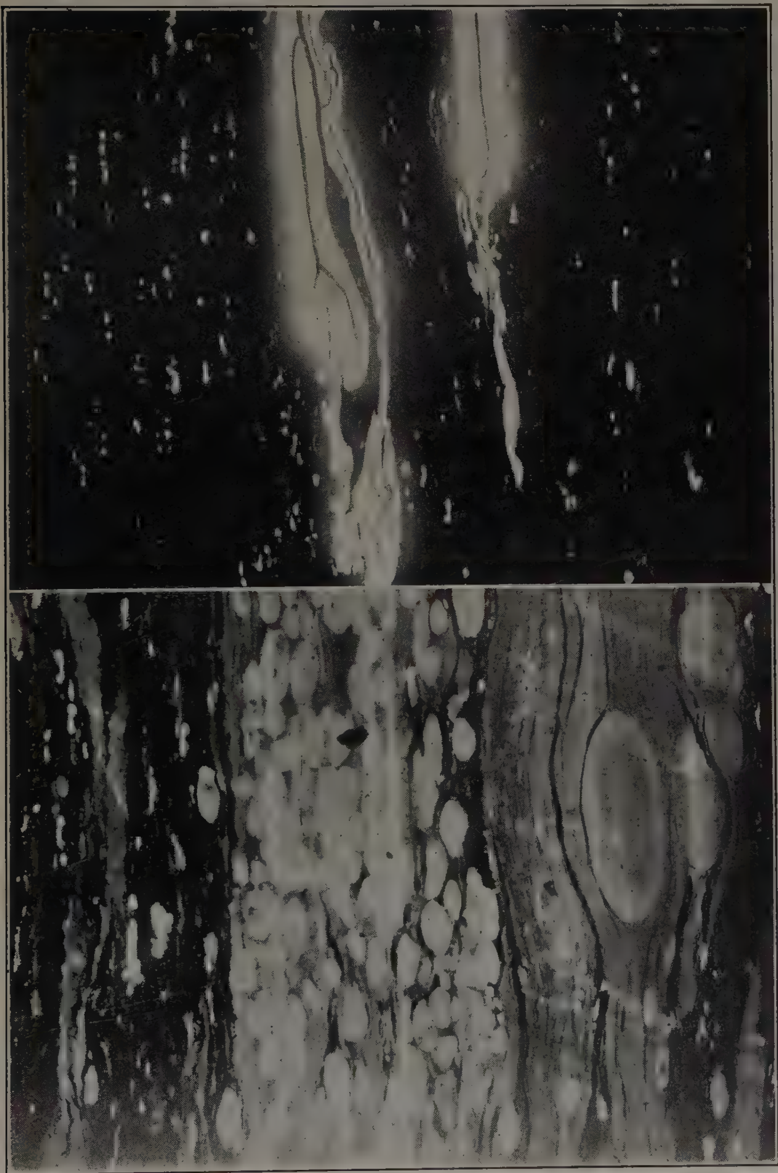


Fig. 21. Fig. 22.

Fig. 21.—Thin cross-section of splint coal, Highsplint bed. × 200. Shows smaller and larger resin globules in anthraxylon.

Fig. 22.—Thin cross-section of splint coal, Highsplint bed. × 100. Shows type of megaspore in splint coals.

Reduced one-third; original magnifications given.

they are denser than ordinary coals of similar chemical composition, they show a much deeper shade of red.

Some evidence of plant structure almost always remains; although as a rule the structure is faint and is often revealed merely as more or less

parallel striae (Fig. 19); nevertheless, definite cell structure is common, as in Fig. 20. Here and there are resinous inclusions of an oval or ovoid globular shape, of greatly varying size, showing that the resins were not absent in the contributing plants (Fig. 21).

Attritus

As already noted, the attritus consists largely of opaque matter in which spores, cuticles, humic matter and resinous particles are embedded. It is, therefore, opaque and dense, and requires exceedingly thin sections to be transparent enough to expose all of the constituents.

Opaque Matter.—The most prominent and important constituent is the opaque matter, also referred to as opaque matrix, black fundamental matter or matrix and residuum.[13] It is of a characteristic nature and is easily recognized wherever present.

In a medium-thin section it appears as a more or less homogeneous mass, but as the section is ground thinner and thinner it is resolved into more or less definite unit particles of varying size, but so extremely variable and irregular in appearance as to defy description. Some of the particles are then transparent; others are still opaque, with all possible degrees of transparency between these two extremes. The best idea of its nature may be obtained from photographs at high magnification, say 1000 dia. It matters little whether the illustrations are taken from our splint coals, the German *Mattkohle*, or the British hards; all are very similar in nature. Fig. 15, of a splint coal from the Highsplint bed in Kentucky, Figs. 16 and 17, of a *Mattkohle* of Flöz Zollverein of the Ruhr, and Fig. 18, of British coals, are good illustrations of the appearance under the microscope.

Humic Matter.—Transparent humic matter—that is, humic matter similar to that which constitutes the major part in the attritus in ordinary coals—is always present in splint coals. It is, however, rather irregularly distributed, and is only sparingly present; but here and there are areas where it forms the predominant constituent (Fig. 12).

Humic matter consists of irregular particles varying greatly in shape and size. Although rounded or ovoid particles are not rare, humic matter is mostly flattened in form and usually of frayed and tattered appearance. The particles are of the same deep red color as the anthraxylon strips, becoming lighter in thinner sections.

Spores.—These are usually present in relatively large numbers, as may be seen from Figs. 9, 11, 13, 14 and 17, yet this is not always the case. In some sections and even in some of these coals as a whole, spores are not

---

[13] J. G. Kellett: The Physical Constitution of Bituminous Coal and Coal Seams. *Coll. Guard.* (1928) **137**, 240–242.

numerous—in other words, numerous spores are not a requisite. Frequently, the spores are much deformed and even torn and macerated. Both megaspores and microspores are present; the latter, however, are more numerous.

A remarkable fact is that the predominant spores of all of the splint coals thus far examined, whether the *Mattkohle* of the Ruhr, the hards of England, or the splint coals of America, are of the salver-shaped type (Figs. 9, 10, 13, 14, 16 and 17). Other types of spores are present, but are few in number. One type, a saw-edged, rather large spore is rather predominant in certain sections. The larger number of megaspores are also of one and the same type—a relatively large spore with appendages (Fig. 22).

The fact that the majority of the spores are of the same kind would indicate that the opaque matter had its origin in a special type of plant. If so, it may be possible that such a plant produced an abundance of a certain product which, during the peat-forming stage, left a residue that gave rise to this opaque matter; or the structure of the whole plant was such that it gave rise to a peculiar form of decomposition resulting in this type of residue.

Cuticles.—These, as a rule, are not abundant in these coals, yet in some sections they may be numerous. When present they are much torn and fragmented. Cuticles are often found associated with definite tissues.

Fusain.—The fusain content varies considerably in the splint coals from different localities and in different sections; as a whole it is not abundant, but when present it is of much the same nature as in other coals.

Mineral Matter.—The amount of mineral matter in splint coal varies considerably, but it is always higher than that in the anthraxylon and lower than that in fusain. Very little, if any, is visible under the microscope. According to British investigators the ash is usually in the form of a fine powder, of a pale gray to almost white color, and consists largely of aluminum silicate. Baranov and Francis[14] give the following analysis for the ash from the durain of the Top Hard bed at East Kirkby colliery: SiO_2, 56.3; Al_2O_3, 39.6; Fe_2O_3, 2.4; CaO_3, 1.0 per cent.; MgO and alkali, a trace. Several analyses from other localities give figures of a similar magnitude.[15]

---

[14] A. Baranov and W. Francis: Banded Bituminous Coal. *Fuel in Science and Practice* (1922) **1**, 219–222.

[15] R. Lessing: Coal Ash and Clean Coal. Canton Lectures. *Trans.* Roy. Soc. Arts. (1926).

N. Simpkin: Note on the Composition of Durain. *Jnl. Soc. Chem. Ind.* (1926) **45**, 76T-78T.

Footnote 15 continued on next page.

Chemical Analysis

The chemical analyses of a few of the American splint coals determined do not differ in the main from those of the ordinary humic coals of similar rank, as shown in the analyses of Table 1, which were made in the laboratories of The Koppers Company.

TABLE 1.—*Analyses of High Splint Coal from Clover Splint[a] Property Compared with Others in the District*

	Six Samples from Clover Splint Property							Fifteen Samples from Same District		
Map Location of Sample	Opening A	Opening B	Opening C	Opening D	Drill-hole Core E	Composite sample 3 Extreme Headings	Average	Average	Maximum	Minimum
Clean coal, inches	59	57	49	55.5	52.0	52.0	54.1	55.5	84	39.5
Volatile matter constituents, per cent	38.10	37.90	37.40	38.20		38.9	38.10	37.10	40.23	34.72
Fixed carbon	58.71	59.59	57.78	58.51		55.3	57.98	58.86	61.34	53.65
Ash	3.19	2.51	4.82	3.29	3.71	3.60	3.52	4.04	7.12	2.76
Sulfur	0.49	0.50	0.55	0.46	0.45	0.51	0.49	0.55	0.69	0.44
Phosphorus	0.019	0.019	0.007	0.006			0.013	0.016	0.044	0.003

[a] A representative analysis of coal from the Clover Splint mine is 39.27 per cent. volatile matter, 57.04 per cent. fixed carbon, 3.69 per cent. ash, 0.55 per cent. sulfur, and 0.024 per cent phosphorus.

SPLINT COAL FROM THE HERNSHAW BED, BOONE COUNTY, KENTUCKY. ANALYSES ON DRY BASIS

Sample No.	V. M., Per Cent.	F. C., Per Cent.	Ash, Per Cent.	S, Per Cent.	C, Per Cent.	H, Per Cent.	O, Per Cent.	N, Per Cent.
12001	32.12	59.11	8.77	0.50	78.57	4.93	6.03	1.20

ORIGIN OF SPLINT COALS

There are evidences to show that splint coals owe their specific and characteristic nature to a certain type of plant—one which, during the formative peat stage, gave rise to larger quantities of opaque matter

H. Winter: Mikroskopische und chemische untersuchungen an Streifenkohlen des Ruhrbezirks. *Glückauf* (1928) **64**, 653–658.

R. Kattwinkel: Untersuchung über die Verkokbarkeit der Gefügebestandteile von bituminosen Streifenkohlen des Ruhrbezirks. *Glückauf* (1928) **64**, 79–83.

W. Rittmeister: Eigenschaften und Gefügebestandteile der Ruhrkohlen. *Glückauf* (1928) **64**, 589–594, 624–637.

than the plants that contributed to the ordinary humic coals. Unfortunately, the origin of the opaque matter is not known; but the fact that, as far as known, the structure and nature of the anthraxylon is similar in all splint coals and that the salver-shaped type of microspores and the winged megaspores which predominate in them are also all of the same kind, lend weight to this theory.

Splint Coals a Type

The characteristic nature of a splint coal persists, regardless of its rank. In other words, specific plant substances gave rise to these coals. Splint coals do not owe their specific characters to metamorphism or dynamic agencies, such as earth movements, earth thrusts and mountain building, to which other coals owe their difference in rank. The American splint coals are of high-volatile bituminous rank; the British durains are of a lower volatile bituminous rank. In the Ruhr district of Germany, where every bed, as far as examined, contains splint coals, these coals increase in rank—namely, from *Gasflammkohle,*[16] *Gaskohle, Kokskohle,* to *Esskohle* and *Magerkohle*—with increasing depth of the beds, parallel with *Glanzkohle* of the same beds. The two components, *Glanzkohle* and *Mattkohle* of the Ruhr coals change chemically and in outward appearance with the degree of incoalation and become more like each other in passing from the lower to the higher rank; but with respect to their microscopic structure they remain distinct.[17] Splint coals, therefore, must be considered as a type of the same order as humic coals, cannel coals and boghead coals.

Misunderstandings of Terms

It has been correctly said[18] that the term "durit" (or "durain") has brought about more confusion in the literature of coal petrography and

[16] After Gruner's classification the Ruhr coals are ranked according to their volatile content, as follows:

Coal	Per Cent.
Anthracite or Magerkohle	5 to 10
Halbfett or Esskohle	10 to 15.5
Fett or Kokskohle	15.5 to 33.3
Backende Gaskohle	33.3 to 40
Gaskohle	40 to 44
Gasflammkohle	44 to 48

R. Potonié: Einführung in die allgemeine Kohlenpetrographie. 1924, Berlin, Bornträger.

[17] H. Winter: *Op. cit.*
W. Rittmeister: *Op. cit.*

[18] H. Bode: Zur Nomenklatur in der Kohlenpetrographie. *Kohle und Erz* (1928) **25**, 699–710.

geology than any other, and that this term is being used with more different meanings by different writers than any other.

One of the main reasons for this confusion and misunderstanding is that the distinction between the two types of attritus—namely, the transparent attritus of the ordinary humic coals (Figs. 3, 4, 5) and the opaque attritus of the splint coals—has not been sufficiently realized by most coal petrographers. As soon as the distinction between a transparent attritus and an opaque attritus has been made, as has been done between the anthraxylon and the transparent attritus or between anthraxylon and the opaque attritus, the confusion will cease, and it will be realized that anthraxylon and transparent attritus constitute ordinary humic coals, and that anthraxylon and opaque attritus constitute splint coals.

Summary

Splint coals, *Mattkohle* or *Streifenkohle* or durit, and hards or dulls or durain represent groups of coals that may be put under one type and should preferably be classed under one name; therefore, since the term "splint coal" has found its way into common use, in the literature and in dictionaries, the word "splint" is used for the whole type in this paper. These coals are present as bands varying from a fraction of an inch to several feet in thickness, interlayered in a bed of ordinary humus coals, or they may constitute the whole bed.

Splint coals are radically different from ordinary coals and are easily distinguishable from them. They are of an irregular lumpy nature, with an irregular rough fracture, and are rarely smooth. They are dull, grayish black in color. They have a granular consistency, are very hard, solid, and tough, and produce very little fines in breaking up. Splint coals are of a relatively higher specific weight, of a different chemical composition and behavior on chemical treatment, and are slightly higher in volatile matter than the ordinary humus coals of the same rank. The average ash content of splint coals varies, but the average is relatively high—higher than that of pure anthraxylon and lower than that of fusain; the ash is light gray, almost white, in color, and consists mainly of aluminum silicate. Splint coals are generally considered to be non-coking.

When carefully observed with the naked eye or with a hand lens, splint coals are seen to be composed largely of a dull granular attritus, intercalcated with numerous thin sheets of anthraxylon. Microscopically considered, the attritus is composed of an opaque matter to which this type of coal owes its characteristic nature. In ordinary thin sections the opaque matter appears under the microscope as a homogeneous opaque mass in which are embedded numerous spores, some cuticles and a certain amount of transparent humic matter, all interlayered by

the sheets of anthraxylon. In exceedingly thin sections the opaque matter is resolved into a heterogeneous mass consisting of more or less definite particles varying greatly in size, form and transparency.

The anthraxylon sheets are relatively thin, varying in thickness from not more than a few millimeters to microscopic size.

The attritus of the splint coals is therefore largely composed of opaque matter, with spores, transparent humic matter, cuticles, and resin particles in minor proportions (Figs. 9 to 22) in sharp contradistinction to the attritus of the ordinary humic coals, which ordinarily is composed largely of transparent humic matter, with spores, cuticles, resins and opaque matter (Figs. 3, 4 and 5) in minor proportions.

Acknowledgments

The writer wishes to express his sincere thanks to Howard N. Eavenson, president of the Clover Splint Coal Co., and to H. J. Rose, assistant director of research, The Koppers Co., for supplying the samples required in the investigation and for furnishing other valuable information.

DISCUSSION

M. R. Campbell, Washington, D. C., inquired as to whether the bands of dark and light coal are due to different plants or to different conditions in the coal bogs.

Dr. Thiessen stated that he believed that the material giving rise to the opaque (in thin section) material came from a given plant or plant constituent, and that different types of coal are due to different types of vegetation.

A Member inquired as to the opaque part under the microscope.

Dr. Thiessen stated that in splint coal the background remains opaque until extreme thinness is reached, when colors from red to grayish appear, depending on the thickness of the section and the intensity of light.

A Member stated that in heating thin sections in the microfurnace, the groundmass changes, becoming more opaque with rising temperature and reaching complete opacity at 300°, the plant remains, cuticles, spores, etc., remaining unchanged. In splint coal the coalification has progressed further than in cannel coal.

Dr. Thiessen pointed out that the opaque particles differ in splint coal in shape and character from those in bituminous coal.

D. White, Washington, D. C. (written discussion).—Durain, one of the four "ingredients" of Paleozoic banded coals, as described by Stopes in 1919, is not only complex in composition—it is anomalous. As originally defined it was to embrace the dull laminated layers, when the latter are relatively thick.

At the Second International Conference on Bituminous Coal held in Pittsburgh in November, 1928, the author of the foregoing paper, who had been studying microscopically a number of British and German coals as well as coals of this country, announced that durain was a peculiar substance not present in any American coal; that we in America had no durain in our coals. Besides noting that this conclusion seemed incredible, I then expressed the view that the opacity of the attrital layers, on account of which they were called durain by him in the European coals, was due

entirely to the stage of progressive carbonization of some of the matter in the attrital (dull) laminae. All of the coals in the higher bituminous ranks become more and more opaque as carbonization advances. More recently the absence of durain from American coals has been reaffirmed.

On reading Dr. Thiessen's paper on splints, I see no reason for a change in my views. It will, I believe, later be fully demonstrated that opaque areas in the attrital bands are more widespread, with further development of opacity as the coals approach and pass into the coking ranks. The American splints which Dr. Thiessen is describing and illustrating with such remarkable distinctness appear to have arrived at the stage of carbonization indicated by 61 or 62 per cent. of fixed carbon in the pure coal. They are nearing the "coking" rank, as that was recognized in "beehive" oven days.

The suggestion that durain may be due to the presence of certain types of spore exines in the coals is confronted by the fact that the spore exines present are themselves still translucent in all parts of the attrital layers. At best, in looking for a special substance, we perhaps can go no further than to assume the presence of some product which, in the process of carbonization, becomes opaque a little earlier than most of the associated matters in the attrital bands. I can not now recall any commodity likely to be present in the Pittsburgh and Harlan County coals that should not be present both in middle and lower Pottsville coals on the one hand, and in other late Pennsylvanian coals on the other. Aside from the Calamariae, which range through the entire Pennsylvanian, there is no known species of land plants that I have found present both in the upper Pottsville coals of Harlan County and in the Monongahela formation, which includes the Pittsburgh bed. Tree ferns, represented by fronds of Pecopteris, are abundant in the Monongahela and Dunkard formations. They might contain large quantities of glandular secretions, but Pecopterids are relatively rare in the upper Pottsville. The Neuropterids and Alethopterids are not only common to both—they range through the entire upper Carboniferous of the Appalachian trough.

Obviously, complete intergradation will be found between the normal humic coals and the splint type, which is composed mainly of attrital matter, including many spores, together with relatively few and thin bands of humified wood, which appear as glistening vitreous lenses and streaks, representing, as Dr. Thiessen points out, the "glance" coal of Germany and the vitrain and bright bands of the "clarain" of the Stopes classification. Some of the splints appear to grade into canneloid phases, though the typical splint appears to be laminated, with considerable fusain—in layers, as usual. The characteristic granular irregular surface may be due to its relatively homogeneous attrital composition, and its grayish fracture, due perhaps in part to the character of the fractured surface, would appear to be a product of the stage of carbonization. What is called durain, with similar opacity and physical aspect, will undoubtedly be found in Mesozoic and Tertiary coals of similar conditions of formation and of the same stages of carbonization.

Commercial Classifications of Coal

By F. R. Wadleigh,* New York, N. Y.

(New York Meeting, February, 1930)

There are in commercial use today in the United States various classifications of coal, each based on one or more characteristics. The bases of these classifications may be described as follows:

General Basis	Examples
1. Geological	By seams and basins (local).
2. Chemical	By constituents, as determined by analysis—ash, volatile, sulfur and heating value.
3. Combustion Characteristics	As free-burning, smokeless, character of ash.
4. Carbonization Characteristics	Coking, noncoking.
5. Physical Characteristics	Splint, soft, hard, block.
6. Geographical	States, Fields, Counties, Districts.
7. Sizes	By dimension or name.
8. Company Names	Producing or sales company names, as "C. C. B. Pocahontas," etc.
9. Trade Names, copyrighted or not	
10. Uses	Gas, steam, coking, locomotive, fuel, bunkers, etc.
11. Use Methods	Stoker coal.
12. Pools (Tidewater Exchanges)	

Analyses	Size	Seams	Railroads
Uses	Structure	Districts	

There are also classifications founded on more than one basis, as follows:

1. Geographical and Geological. By locality of mines (State, District, County) and seam.
   Examples: Cambria County, Miller Seam, New River, Sewell Seam.
2. Geographical and Chemical. Alabama medium volatile.
3. Combustion Characteristics and Ash Content. Free-burning, red ash (anthracite).
4. Carbonization Characteristics and Analysis. Noncoking, high-volatile, low sulfur.
5. Physical and Carbonization Characteristics. Splint, noncoking.
6. Size and Use. ¾-in. gas coal; domestic sizes (prepared sizes); steam sizes.
7. Company Name and Seam. Consolidation Co., Elkhorn.
8. Size and Analysis. Low-volatile run-of-mine; low-volatile slack.
9. Sizes—size of screen and name. 6 by 3-in. lump.
10. Chemical and Use. High-volatile steam.
11. Geographical and Use. Youghiogheny gas.
12. District, Size and Use. Pittsburgh gas 1¼-in. lump.

* Consulting Engineer.

13. Pools, as classified by the Tidewater Exchanges. Use, chemical characteristics, size, seam, district, railroads, structure. Tidewater port.
14. Geographical and Size. Westmoreland ¾ inch.
15. Seam and Size. Chilton lump.
16. Trade name and Use. Roda Gas.
17. Company name and size (anthracite).
18. Trade name and district, as "Admiralty New River."

The most commonly used general classification is the following:

	Per Cent. Volatile
Anthracite	up to 10
Semianthracite	10 to 12
Semibituminous	15 to 25
Bituminous medium volatile	25 to 32
Bituminous high volatile	32 and over
Subbituminous	
Lignite	
Cannel	

In order to illustrate the use of the different coal classification bases just listed, there are given:

1. Table of market descriptions as listed for price quotations.
2. Tidewater Coal Exchanges Classification by pools.
3. Group classification of U. S. Eastern coals available for export.
4. Ore and Coal Exchange Classification of coals at Lake Erie ports for water shipment.
5. Commercial classifications of British, German, Belgian and French coals.

Classification and Description of Coals for Price Quotations

The following table of market descriptions was prepared for use by a trade publication in giving weekly spot price quotations; it is intended to include all coals of the United States except those on the Pacific Coast and in Texas.

Spot Prices of Coal, f.o.b. Mines

Anthracite, Prices, per Long Ton; Bituminous, Tidewater Pier and Bunker Prices, per Long Ton, f.o.b.
Piers—f.a.s. Bunkers
May 26 June 2 June 9

Description of Coal	Market Quoted
Low-volatile run of mine Navy standard	Pool 1, Hampton Roads, New York, Philadelphia
Other low-volatile run of mine	Pool 2, Hampton Roads; Pool 11, New York, Philadelphia, Baltimore
Low-volatile nut and slack	Pool 3, Hampton Roads
High-volatile run of mine	Pools 5, 6, 7, Hampton Roads
High-volatile lump	Pools 41-43, Hampton Roads
Superior low-volatile run of mine	Pool 9, New York, Philadelphia, Baltimore
High-grade low-volatile run of mine	Pool 10, New York, Philadelphia, Baltimore

Spot Prices of Coal, f.o.b. Mines.—(Continued)

SPOT PRICES, NET TONS, F.O.B. MINES

DESCRIPTION OF COAL	MARKET QUOTED
Eastern Coals	
Low-volatile Navy standard	Pool 1, Hampton Roads, New York, Philadelphia, Baltimore
Other low-volatile	Pool 2, Hampton Roads
Nut and slack	Pool 3, Hampton Roads
Superior low volatile	Pool 9, New York, Philadelphia, Baltimore
High-grade low-volatile	Pool 10, New York, Philadelphia, Baltimore
Other low-volatile	Pool 11 and Pool 18, New York, Philadelphia, Baltimore
Low-volatile, r.o.m., Hampton Roads, Boston on cars.	
Low-volatile, r.o.m., Penna., all rail Boston f.o.b. mines.	
Pocahontas	
New River, W. Va.: lump, egg, nut, mine run and slack.	Chicago, Cincinnati, Cleveland, Columbus, Detroit, Upper Lake Docks
Medium-volatile (25–31 per cent.), Pennsylvania	New York, Philadelphia, Baltimore
High-volatile, Pennsylvania	
Pittsburgh Gas: ¾-in. lump, 1¼-in. lump, r.o.m. slack	Pittsburgh
Pittsburgh Steam: ¾-in. lump, 1¼-in. lump, r.o.m., slack	Pittsburgh
Panhandle Steam: lump, r.o.m., slack, domestic lump	Pittsburgh
Westmoreland Gas: ¾-in. lump, m.r., slack	Pittsburgh
Bessemer (thin vein): ¾-in. lump, 1¼-in. lump, r.o.m., slack	Pittsburgh
West Virginia	
Fairmont-Morgantown Steam: ¾-in. lump, r.o.m., slack	New York, Philadelphia, Baltimore.
Fairmont Gas: ¾-in. lump, r.o.m., slack.	New York, Philadelphia, Baltimore.
Kanawha-Logan-Thacker: lump, 2-in. lump, egg, r.o.m. gas, r.o.m. steam, slack, r.o.m. by-product.	Columbus, Cincinnati, Chicago, Cleveland, Upper Lake Docks.
Eastern Kentucky and Eastern Tennessee: block, egg, r.o.m., steam, 2-in. lump, slack.	Cincinnati, Chicago, Cleveland, Upper Lake Docks, Louisville.
Ohio	
Hocking Pomeroy: lump, r.o.m., slack	Cleveland, Columbus, Chicago.
Pittsburgh No. 8: lump, r.o.m., slack.	Cleveland, Columbus, Chicago.
Indiana	
Indiana No. 4: lump, egg and nut, m.r., slack	Chicago
Indiana No. 5: lump, egg and nut, m.r., slack	Chicago
Western Kentucky: 6-in. lump, 3-in. lump, egg, nut, m.r., 1½-in. screening	Chicago, Louisville, St. Louis.

Spot Prices of Coal, f.o.b. Mines.—(Continued)

Description of Coal	Market Quoted
Illinois	
Southern Illinois: lump furnace, small egg, stove, chestnut, pea, carbon, 1¼-in. screenings, 2-in. screenings, m.r.	Chicago
Central Illinois: domestic sizes, steam egg, m.r., screenings	Chicago
Belleville (standard): lump, m.r., steam egg, steam nut, screenings	Chicago, St. Louis.
Northern Illinois: lump, egg, m.r., screenings	Chicago
Alabama	
Cahaba: m. r., washed, lump	Birmingham
Carbon Hill: m.r., washed, lump	Birmingham
Black Creek: m.r., washed, lump	Birmingham
Big Seam: m.r., washed, lump	Birmingham
Pratt: r.o.m., washed	Birmingham
Corona: r.o.m., washed, lump	Birmingham
Montevallo: lump	Birmingham

Kansas
- Cherokee Deep Shaft: lump, nut, m.r., screenings.

Iowa
- Des Moines District: 8-in. lump and egg, screenings.
- Centerville: 8-in. lump, screenings.
- Arkansas: semianthracite, lump, nut, slack.
- Paris: lump, Spadra deep shaft, grate and egg, No. 4, range, chestnut.
- Missouri: Novinger, lump, nut, m.r., screenings.
- Huntsville: lump, m.r., screenings.

Colorado
- Walsenberg, Canon City: bituminous
- Crested Butte, Maitland, Routt Co.: lump, nut, chestnut; Horace furnace and baseburner anthracite, special brooder mixture, chestnut.
- Trinidad: all prepared sizes.
- Elk Mountain Furnace: baseburner, chestnut.

Oklahoma
- McAlester: lump, nut, chestnut, slack.
- Wilburton: lump, nut, chestnut, slack.
- Henrietta and Dewar: lump, nut, slack.
- Magic City (Tulsa): lump, nut.
- Superior (LeForge County): semianthracite lump.

Wyoming
- Rock Springs: lump and stove, nut, m.r., slack.

Spot Prices of Coal, f.o.b. Mines.—(Continued)

COKE

Beehive

Furnace, f.o.b. ovens—Connellsville, New River, Wise County (Va.), Alabama.
Foundry, f.o.b. ovens—Connellsville, New River, Wise County (Va.), Alabama.

By-product

F.o.b. Chicago: domestic, foundry.
Duluth, Birmingham or New York: domestic, foundry.
Boston: foundry and crushed, f.o.b. ovens, Everett or W. Va. delivered.
Philadelphia, Buffalo, Newark, Indianapolis, St. Louis, Ashland, Ky. and Portsmouth, Ohio: domestic, foundry.

TIDEWATER COAL EXCHANGES, POOL CLASSIFICATIONS

The so-called "Pool" classifications were put into operation on Aug. 1, 1917, by the Tidewater Coal Exchange, which was organized as a war measure to handle all bituminous coal shipped to Atlantic tidewater ports. The main purpose of the Exchanges was to assist the railroads in handling tidewater coal by expediting movements and saving in use of transportation facilities.

After the war (May 1, 1920) the Tidewater Coal Exchange, Inc. was created to handle tidewater shipments over certain railroads at Baltimore, Philadelphia and New York, while separate exchanges were later organized at Newport News by the C. & O. R. R. (July 8, 1920), at Lamberts Point by the N. & W. R. R. (Dec. 1, 1920) and at Sewalls Point by the Virginian R. R. (July 1, 1921); in cooperation with shippers of coal on the railroads named.

The classifications adopted by the Exchanges, dividing the coals into pools entirely for commercial use, were based primarily on quality (as shown by analysis and heating value determination) and uses; sizes, structure, seams, producing districts and railroads and tidewater ports also entered into the classification, of which a complete list is given below.

In the Tidewater Exchange classifications, not only were the coals classified but all mines were classified by pools. While the Tidewater Exchanges have all gone out of existence, some of the pool classification numbers are still used by coal trade publications, some individuals and coal sales companies, in giving out market quotations.

The pool classifications served their general purpose, but they were, with some exceptions, often inadequate, incorrect, and misleading, and were the source of considerable trouble and litigation. At the same time these pools represented the first systematic attempt at commercial classification in the United States and may be used as a basis for further efforts, for which purpose they are given in full.

The first pool classification of the original Tidewater Coal Exchange made effective in July, 1917, was as follows:

Consigning Pool Numbers

Hampton Roads

Originating on Chesapeake & Ohio and Norfolk & Western Railways.	
American standard low-volatile, r.m.	1
Other low-volatile steam, r.m.	2
Low-volatile slack	3
Low-volatile by-product, r.m.	4
Taggart seam gas, r.m.	40
Splint lump	41
Other high-volatile gas, r.m.	5
Other high-volatile steam, r.m.	6
Other high-volatile by-product, r.m.	7
Originating on Virginian Railway.	
American standard low-volatile, r.m.	1
Pocahontas seam	42
Other low-volatile, r.m.	2
Low-volatile slack	3
High-volatile steam, r.m.	6
Eagle seam low-volatile, r.m.	8

New York, Philadelphia and Baltimore

Originating on New York Central R. R.	
American standard low-volatile, r.m.	1
Other B seam, r.m.	9
C prime and D seams, r.m.	10
Other low-volatile, r.m.	11
By-products, r.m.	12
Originating on Buffalo, Rochester & Pittsburgh Railroad.	
American standard low-volatile, r.m.	1
By-product, r.m.	12
Coal loaded otherwise than through regular tipples having track connections, r.m.	13
Pittsburgh seam, r.m.	14
Other coals, r.m.	15
Slack, r.m.	16
Originating on Pittsburg, Shawmut & Northern, Pittsburg & Shawmut and Buffalo & Susquehanna Railroad, r.m	17
Originating on Pennsylvania Railroad.	
American standard low-volatile, r.m.	1
Other B seam (including East Broadtop), r.m.	9
C prime	
C prime, D and E seams, r.m.	10
Other low-volatile, r.m.	11
Coal loaded otherwise than through regular tipples having track connections, r.m.	13
Huntingdon and Broadtop, r.m.	18
Low-volatile by-product, r.m.	19
Low-volatile slack	20
High-volatile by-product, r.m.	21
Gas, ¾-inch	30
Gas, slack	32
High-volatile steam ¾-inch	33

Consigning Pool Numbers.—(Continued)

High-volatile steam, r.m.	34
High-volatile steam slack	35
Originating on Baltimore & Ohio and Western Maryland (Inc. P. & L. E.).	
American standard low-volatile, r.m.	1
Coal loaded otherwise than through regular tipples having track connections, r.m.	13
C prime seam, r.m.	22
D and E seams, r.m.	23
Other low-volatile, r.m.	24
High-volatile by-product, r.m.	25
Low-volatile by-product, r.m.	26
Youghiogheny gas ¾-inch	36
Marion Co. gas ¾-inch	37
Marion Co. gas, r.m.	38
Other high-volatile steam ¾-inch	33
Other high-volatile steam, r.m.	34
Other high-volatile slack	35
Originating on Cumberland & Pennsylvania and Georges Creek & Cumberland.	
American standard low-volatile, r.m.	1
Other Pittsburgh and Tyson vein, r.m.	39
Other low-volatile, r.m.	24

The Tidewater Coal Exchange classifications were revised on March 18, 1920, as follows:[1]

Coals on the Norfolk & Western, Chesapeake & Ohio and Virginian Railways and Connections

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
Other high-grade Pocahontas-flat Top, Tug River and New River low-volatile	r.m.	2
Pocahontas No. 3 seam low-volatile (on Virginian Ry.)	r.m.	2-P
Low-volatile slack	slack	3
High-volatile gas	r.m.	5
High-volatile steam	r.m.	6
High-volatile by-product	r.m.	7
Eagle seam (on Virginian Ry.)	r.m.	8
Coal loaded otherwise than over regular tipples having track connections (wagon mines)	r.m.	13
High-volatile gas slack	slack	32
High-volatile steam slack	slack	35
Elkhorn seam gas	r.m.	39
Taggart seam (on N. & W. Ry.)	r.m.	40
High-volatile steam lump	lump	41
Medium grade Pocahontas-flat Top, Tug River and New River low-volatile	r.m.	42

[1] U. S. Bur. Mines *Bull.* 230 (N. H. Snyder) contains a chapter on Tidewater Pool Classifications, which gives much of the information detailed in the following pages.

Coals on the Norfolk & Western, Chesapeake & Ohio and Virginian Railways and Connections.—(Continued)

	Size	Pool No.
High-volatile gas lump	lump	43
Pocahontas-Flat Top, Tug River & New River low-volatile	lump	44
High-volatile steam egg	egg	51
High-volatile gas egg	egg	53
Pocahontas-Flat Top, Tug River and New River egg	egg	54
Medium grade high-volatile steam	r.m.	56
High-volatile steam nut	nut	61
High-volatile gas nut	nut	63
Pocahontas-Flat Top, Tug River and New River nut	nut	64
Pocahontas-Flat Top, Tug River and New River pea	pea	84

Note.—All mines shown herein under Pools 41, 43 and 44 for Lump size coal must ship their Egg size coal to Pools 51, 53 and 54, and their Nut size coal to Pools 61, 63 and 64, respectively; and Pool 44 mines must ship their Pea size coal to Pool 84, as Pools 41, 43 and 44 are reserved exclusively for Lump size coal and no smaller sizes of coal will be accepted in these pools.

Coals on Baltimore & Ohio, Western Maryland, Pittsburgh & Lake Erie, Cumberland Valley, Cumberland & Pennsylvania Railroads and Connections

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
Somerset "C" prime	r.m.	9
Superior low-volatile	r.m.	10
Good low-volatile	r.m.	11
Fair low-volatile	r.m.	18
Low-volatile slack (From Pools 1, 9, 10 and 11)	slack	20
Low-sulfur gas (Pittsburgh district)	r.m.	31
	slack	32
High-volatile steam (Fairmont-Pittsburgh seam)	¾-in.	33
	r.m.	34
	slack	35
Low-sulfur gas (Pittsburgh district)	¾-in.	36
Marion Company gas	¾-in.	37
	r.m.	38
High-volatile Northern West Va. (Kittanning and Sewickley seams)	¾-in.	43
	r.m.	44
	slack	45
Low-sulfur illuminating gas (Westmoreland-Youghiogheny districts)	¾-in.	60
	r.m.	61
	slack	62
Superior low-volatile (mines on Navy supplementary list)	r.m.	71

Coals on Pennsylvania Railroad and Connections, Including East Broad Top R. R. & Coal Co., Ligonier Valley & Monongahela Railroad

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
High-grade South Fork-Miller Vein-E. B. T. R. R. & C. Co., under 26 per cent. volatile	r.m.	9
Superior low-volatile steam	r.m.	10
Good low-volatile steam	r.m.	11
Medium-volatile (25–30%)	r.m.	15
Fair low-volatile steam	r.m.	18
Low-volatile slack from Pools 1, 9, 10, 11	slack	20
High-volatile by-product	r.m.	21
Gas coal, Pittsburgh district	¾-in.	30
	r.m.	31
	slack	32
High-volatile steam	¾-in.	33
	r.m.	34
	slack	35
Greensburg Basin	¾-in.	39
	r.m.	40
Illuminating gas coal (Max. sulfur 1¼%) Westmoreland-Irwin Basin district	¾-in.	60
	r.m.	61
	slack	62
Superior low-volatile (mines on Navy supplementary list)	r.m.	71

Coals on New York Central; Buffalo, Rochester & Pittsburgh; Cambria & Indiana; Pittsburgh & Susquehanna Railroads and Connections

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
Good B seam (N. Y. C., C. & I., P. & S.)	r.m.	4
C. and D. seams (N. Y. C., B. R. & P., C. & I., P. & S.)	r.m.	10
Other good low-volatile (N. Y. C., C. & I., P. & S.)	r.m.	11
By-product	r.m.	12
Pittsburgh seam (B. R. & P., P. S. & N., B. & S., P. & Shaw)	r.m.	14
Fair low-volatile steam and R. R. fuel (N. Y. C., C. & I., P. & S.)	r.m.	18
Slack	slack	20
Superior low-volatile (mines on Navy supplementary list)	r.m.	71

Tidewater Coal Exchange, Inc., put into effect on March 15, 1921, the following classifications:

Description of Pools

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
Superior low-volatile (N. Y. C., P. & S., C. & I.)	r.m.	4
Superior low-volatile, all other roads	r.m.	9
High-grade low volatile, all roads	r.m.	10
Low-volatile, all roads	r.m.	11
Superior medium volatile, all roads	r.m.	12
Medium volatile, all roads	r.m.	14
Other medium volatile coal, all roads	r.m.	15
Other low-volatile coal, all roads	r.m.	18
Slack coal from low-volatile mines (in Pools 1-4-9-10-11)	slack	20
High-volatile steam coal (Fairmont & Pittsburgh Districts)	r.m.	21
	¾″ lump	30
Slack coal from Pools 34-38-44-54-64	r.m.	31
Fairmont, low-sulfur gas coal	slack	32
	¾″ lump	33
Greensburg, high-volatile	r.m.	34
	slack	35
High-volatile steam coal (all seams)	¾″ lump	37
	r.m.	38
Superior high-volatile coal (Sewickley, Kittanning, Wainesburg & Freeport seams)	¾″ lump	39
	r.m.	40
	¾″ lump	43
	r.m.	44
Westmoreland Irwin Basin and Youghiogheny low-sulfur illuminating gas coal	¾″ lump	53
	r.m.	54
	¾″ lump	60
	r.m.	61
Pittsburgh and Redstone seams, hard-structure gas coal	¾″ lump	63
	r.m.	64
Mines on Navy supplementary list	r.m.	71

The Newport News Coal Exchange, Inc. Classification, as revised, effective March 1, 1922, was as follows:

Description of Classifications

	Size	Pool No.
Standard New River low-volatile	r.o.m.	1
Other high-grade New River low-volatile	r.o.m.	2
Standard and other high-grade New River low-volatile	Lump and egg	44
	Nut and slack	3
Medium grade New River low-volatile	r.o.m.	42
Meadow River and other low-volatile coals	r.o.m.	12
Note.—Analysis on this coal will compare favorably with Pools 1 and 2 but will average a much higher per cent. of slack and fines.		
Standard high-volatile gas	r.o.m.	15
Standard high-volatile gas and by-product	Lump and egg	45
Standard high-volatile gas and by-product	Nut and slack	37

Description of Classifications.—(Continued)

	Size	Pool No.
High-volatile gas	r.o.m.	5
	Lump and egg	43
	Nut and slack	32
Standard high-volatile by-product	r.o.m.	7
Standard high-volatile firm-structure splint	r.o.m.	16
	Lump and egg	46
	Nut and slack	36
High-volatile splint	r.o.m.	6
	Lump and egg	41
	Nut and slack	35
Medium grade high-volatile coals	r.o.m.	56

The Lamberts Point Coal Exchange coal classifications, effective December 1, 1920, were as follows:

Description of Classifications

	Size	Pool No.
American standard low-volatile (mines on Navy acceptable list)	r.m.	1
Other high-grade Pocahontas-Flat Top and Tug River low-volatile	r.m.	2
Low-volatile slack	slack	3
Medium grade Pocahontas-Flat Top and Tug River low-volatile	r.m.	4
High-volatile gas	r.m.	5
High-volatile steam	r.m.	6
High-volatile by-product	r.m.	7
High-volatile gas slack	slack	32
High-volatile steam slack	slack	35
Taggart seam	r.m.	40
High-volatile steam lump	lump	41
High-volatile gas lump	lump	43
Pocahontas-Flat Top and Tug River low-volatile lump	lump	44
High-volatile steam egg	egg	51
High-volatile gas egg	egg	53
Pocahontas-Flat Top and Tug River egg	egg	54
Medium grade high-volatile steam	r.m.	56
High-volatile steam nut	nut	61
High-volatile gas nut	nut	63
Pocahontas-Flat Top and Tug River nut	nut	64
Pocahontas-Flat Top and Tug River pea	pea	84

The Sewalls Point Exchange put into effect March 15, 1921, the first commercial classification of coal based on standards established by tests of the U. S. Government, in cooperation with the U. S. Bureau of Mines. This classification, however, applied only to coals shipped over the Sewalls Point piers of the Virginian Railway.

Of these coals, 90 per cent. was classed as low volatile, 5 per cent. medium volatile (not over 32.50 per cent.) and 5 per cent. high volatile.

The pools, with their descriptions and specifications were as follows:

Sewalls Point Standard Pools

	Pool No.	Size	Specification
Sewalls Point standard low-volatile..........	1	r.o.m.	1
	2	r.o.m.	2
	4	r.o.m.	4
Low-volatile slack........................	3	slack	None
Sewalls Point standard medium volatile......	8	r.o.m.	8

Specification 1.—Low-volatile run-of-mine coal from mines which on the average of all tests of the Sewalls Point Coal Exchange for a given mine shows more than 15,500 B.t.u. per pound of moisture-free and ash-free coal and which maintain a standard of ash in the dry coal analysis of not more than 7.50 per cent. of ash as determined by test of the Exchange. Coal with respect to size, coarseness and preparation thereof shall not be detrimental to the pool.

Specification 2.—Low-volatile run-of-mine coal from mines which on the average of all tests of the Sewalls Point Coal Exchange for a given mine shows more than 15,500 B.t.u. per pound of moisture-free and ash-free coal and which maintain a standard of ash in the dry coal analysis of not more than 8.50 per cent. of ash as determined by test of the Exchange. Coal with respect to size, coarseness and preparation thereof shall not be detrimental to the pool.

Specification 4.—Low-volatile run-of-mine coal from mines which on the average of all tests of the Sewalls Point Coal Exchange for a given mine shows more than 15,500 B.t.u. per pound of moisture-free and ash-free coal and which maintain a standard of ash in the dry coal analysis of more than 8.50 per cent. of ash but not more than 12.50 per cent. of ash as determined by test of the Exchange. Coal with respect to size, coarseness and preparation thereof shall not be detrimental to the pool.

Specification 8.—Medium volatile run-of-mine coal from mines which on the average of all tests of the Sewalls Point Coal Exchange for a given mine shows not less than 15,400 B.t.u. per pound or more than 32.50 per cent. volatile both on the moisture-free and ash-free basis and which maintain a standard of ash in the dry coal analysis of not more than 8.50 per cent. of ash as determined by test of the Exchange. Coal with respect to size, coarseness and preparation thereof shall not be detrimental to the pool.

All other coal which does not meet the specifications of the pools as enumerated and set out above must be shipped as unclassified coal.

Standard Analyses

Dry Coal						
Pool	Moisture, Per Cent.	Volatile, Per Cent.	Fixed Carbon, Per Cent.	Sulfur, Per Cent.	Ash, Per Cent.	B.t.u.
1		18.50	75.50	0.75	6.00	14,700
2	2.50	17.50	74.50	0.75	8.00	14,400
4		15.00	73.00	0.75	10.00	14,100
8		28.75	63.25	0.90	8.00	14,300

In view of the fact that there will always be some variation in coal from car to car, which cannot be controlled, the closeness with which the standard analyses will be approximated in deliveries out of the pools depends somewhat upon the size of the deliveries. In large cargoes or a number of small deliveries over a period in which an average run of the pools will be had, the standard analyses will be very closely approximated.

In 1921, the following group classification of coals shipped to Atlantic Tidewater ports for exports were made by F. R. Wadleigh and published in the *Black Diamond;* it was afterwards published in the *Keystone Coal Catalog* of 1922 and has appeared in each succeeding issue, including 1927.

This classification placed the Tidewater coals in six groups, based on production locality; in each group the coals were classed according to percentage of volatile matter—low, medium and high—and a range of analyses (content) was given for each class. Tidewater shipping ports were given for each group with the corresponding pool classification.

Group I.—Anthracite classed by sizes, with range of analyses for each size.

Group II.—Pennsylvania Coals: Shipping Ports, New York, Philadelphia, Baltimore.

LOW-VOLATILE COALS, 15 to 25 per cent. Included in the following pools: 1, 9, 71, 10, 11, 18.

Analyses

Moisture, per cent	1 to 3
Volatile matter, per cent	15 to 25
Fixed carbon, per cent	78 to 61
Ash, per cent	6 to 11
Sulfur, per cent	0.7 to 2.50
B.t.u., dry	13,500 to 14,600
Ash fusing temp., °F	2,200 to 2,900

MEDIUM-VOLATILE COALS, 25 to 30 per cent. Also from the Central Pennsylvania district. Included in pools 15, 12 and 14.

Analyses

Moisture, per cent	1 to 3.5
Volatile matter, per cent	25 to 29
Fixed carbon, per cent	66 to 58
Ash, per cent	7 to 11
Sulfur, per cent	0.7 to 1.50
B.t.u., dry	14,400 to 13,600
Ash fusing temp., °F	2,720 to 2,190

HIGH-VOLATILE COALS, 30 to 37 per cent. From several counties in Western Pennsylvania, mainly from Pittsburgh, Youghiogheny and Westmoreland districts, and from the Pittsburgh bed. Pool classifications: 39, 40, 54, 60, 61, 64, 30, 31.

Analyses

Moisture, per cent	1 to 4
Volatile matter, per cent	30 to 37
Fixed carbon, per cent	62 to 57
Ash, per cent	6.5 to 10.5
Sulfur, per cent	0.7 to 2.0
B.t.u., dry	14,400 to 13,500
Ash fusing temp., °F	2,100 to 2,600

Standard Gas and Coking Coals

	Westmoreland Gas	Youghiogheny Gas	Connellsville Coking
Moisture, per cent	2.15	1.15	1.26
Volatile matter, per cent	33.80	35.10	31.80
Fixed carbon, per cent	57.50	57.65	59.79
Ash, per cent	6.55	6.10	7.16
Sulfur, per cent	0.97	0.78	0.53
B.t.u., dry	14,301	14,151	0.024

Group III.—Northern West Virginia and Maryland Coals. Shipping Ports, Baltimore and Philadelphia.

LOW-VOLATILE COALS, 15 to 25 per cent. Pool Classifications: 1, 9, 11, 18.

Analyses

Moisture, per cent	1 to 3
Volatile matter, per cent	15 to 25
Fixed carbon, per cent	77 to 60
Ash, per cent	7 to 12
Sulfur, per cent	0.7 to 2.50
B.t.u., dry	13,700 to 14,600
Ash fusing temp., °F	2,400 to 3,100

HIGH-VOLATILE COALS, 30 to 38 per cent. Pool Classifications: 37, 38, 54, 64.

Analyses

Moisture, per cent	1.5 to 3.50
Volatile matter, per cent	40.00 to 38.00
Fixed carbon, per cent	48.00 to 56.00
Ash, per cent	6.5 to 11.00
Sulfur, per cent	0.9 to 3.50
B.t.u., dry	13,200 to 14,250
Ash fusing temp., °F	2,010 to 2,410

Standard Fairmont Gas Coals

Moisture	1.15
Volatile matter	36.60
Fixed carbon	55.88
Ash	6.27
Sulfur	0.876
B.t.u., dry	14,300

Group IV.—Shipping Ports: Norfolk, Va., Southern West Virginia Coals; Lamberts Point, Norfolk & Western Railway; Sewalls Point, Virginian Railway; Newport News, Va., Chesapeake & Ohio Railway.

LOW-VOLATILE COALS, 16 to 25 per cent. Pool classification: 1, 2, 3, 4.

Analyses

Moisture, per cent	1 to 3
Volatile matter, per cent	16 to 25
Fixed carbon, per cent	79 to 64
Ash, per cent	4 to 8
Sulfur, per cent	0.5 to 1.25
B.t.u., dry	14,400 to 15,000
Ash fusing temp., °F	2,200 to 2,900

Coal Beds—Principal

Pocahontas No. 3, 4, 5, 6. Pocahontas-McDowell, Mercer, Wyoming and Raleigh counties, West Virginia
Davy-Sewell
Welsh
War Creek
Sewell, New River-Fayette, Raleigh and Wyoming counties, West Virginia
Beckley
Fire Creek

Analyses

Coals as shipped; averages of 2369 analyzed and sampled by U. S. Bureau of Mines.

	Pocahontas	New River
Moisture, as received, per cent	2.64	2.58
Volatile matter, dry basis, per cent	18.56	20.32
Fixed carbon, per cent	74.80	74.10
Ash, per cent	6.14	5.58
Sulfur, per cent	0.68	0.82
B.t.u.	14,800	14,825

Ash-fusing Temperatures

Bureau of Mines reports, mine samples

	Pocahontas	New River
No. of mines	73	38
No. of samples	269	148
Fusing temp., °F	2100 to 3010	2080 to 3000
Average	2440	2540

Medium-volatile Coals, 25 to 30 per cent.

Analyses

Moisture, per cent	1.00 to 3.50
Volatile matter, per cent	25.50 to 29.00
Fixed carbon, per cent	66.00 to 62.00
Ash, per cent	4.00 to 8.00
Sulfur, per cent	0.7 to 1.6
B.t.u., dry basis	14,400 to 14,700
Ash fusing temp., °F	2,360 to 2,960
Phosphorus	0.005 to 0.010

High-volatile Coals, 30 to 38 per cent. Pool classifications: 5, 6, 7, 56.

Analyses

	Gas Coals	Splint Coals
Moisture, per cent	1.0 to 3.5	
Volatile matter, per cent	32 to 36	34 to 38
Fixed carbon, per cent	57 to 62	56 to 54
Ash, per cent	4 to 9	7 to 10
Sulfur, per cent	0.6 to 1.60	0.5 to 1.4
B.t.u., dry	13,900 to 14,600	13,700 to 14,300
Ash fusing temp., °F	2,200 to 2,850	2,410 to 3,010

Typical Analyses, Coal As Shipped

	Best Kanawha Gasmaking	Best Locomotive	Best By-product Coking	Best Splint
Moisture, as received, per cent	2.07	1.60	3.72	1.47
Volatile matter, dry, per cent	34.20	32.22	29.38	36.23
Fixed carbon, dry, per cent	60.12	60.62	65.47	56.53
Ash, dry, per cent	5.68	6.24	5.14	7.24
Sulfur, dry, per cent	0.60	1.40	0.9	0.57
Phosphorus, dry, per cent		0.0045	0.0045	
B.t.u.	14,354	14,480	14,370	14,303
Ash fusing temp., mine sample, °F		2,690		2,960

Group V.—Southwest Virginia (not West Virginia) and Eastern Kentucky Coals. Shipping Ports: Norfolk, Va., Lamberts Point, Norfolk & Western Railway; Newport News, Chesapeake & Ohio Railway; Charleston, S. C. Small tonnages are also shipped from Savannah, Ga., and Jacksonville, Fla.

The coals in this group are all medium and high-volatile, from 28 to 40 per cent., but differ considerably in structure and character. Among them are found some of the best gasmaking and by-product coking coals in the world; many of them are excellent steam and locomotive fuels; others are much used as domestic fuels.

Analyses

Moisture, per cent	1.5 to 4.0
Volatile matter, per cent	28 to 40
Ash, per cent	3 to 10
Sulfur, per cent	0.5 to 1.7
B.t.u., dry	13,450 to 14,500
Ash fusing temp., °F	2,180 to 2,940

Typical Analyses, Coal as Shipped

	Elkhorn Coking	Clinchfield Steam	Harlan Gas
Moisture, as received, per cent	2.46	2.65	2.69
Volatile matter, dry, per cent	31.97	35.50	35.70
Fixed carbon, dry, per cent	61.62	57.45	59.97
Ash, dry, per cent	7.41	7.05	4.33
Sulfur, dry, per cent	0.44	0.68	0.85
B.t.u., dry		14,100	14,344
Ash fusing temp., °F	2,470	2,420	2,700

Group VI.—Alabama Coals.

Shipping Ports: Mobile, Pensacola, New Orleans, Jacksonville. Exports of these coals are as yet small and most of the shipments have gone to Cuba, West Indies, Mexico and small tonnage to South America.

Analyses

Moisture, per cent	2 to 5
Volatile matter, per cent	26 to 35
Fixed carbon, per cent	60 to 53
Ash, per cent	6.5 to 12
Sulfur, per cent	0.70 to 2.7
B.t.u., dry	14,400 to 12,840
Ash fusing temp., °F	2,180 to 2,950

Typical Analyses, Coal as Shipped

	Best Black Creek Steam	Cahaba Steam	Pratt Coking	Blue Creek Coking
Moisture, as received, per cent	1.30	3.50	2.70	2.85
Volatile matter, dry, per cent	33.70	32.50	27.50	26.60
Fixed carbon, dry, per cent	61.25	54.69	63.25	60.90
Ash, dry, per cent	3.75	9.40	6.55	9.65
Sulfur, per cent	1.15	1.00	1.20	0.70
B.t.u., dry	14,400	13,200	14,150	13,500
Ash fusing temp., °F	2,530	2,420	2,430	2,690

Lake Coals

In addition to the Tidewater Coal Exchanges classification, the coals shipped from the Lake Erie ports to the Northwest and Canada by water were also classified, using much the same bases as the Tidewater Exchanges.

The Lake Classification was put into effect in June, 1917, by the Lake Erie Bituminous Coal Exchange, afterwards (1918) the Ore and Coal Exchange; this Exchange has continued in existence and is now operated by the railroads at their own expense but without pool classification.

Lake coal comes from six states: Pennsylvania (anthracite and bituminous), West Virginia, Kentucky, Tennessee, and Ohio, and from the following seams:

Pennsylvania:
- Anthracite, all seams
- Bituminous:
  - Brookville
  - Freeport (3)
  - Kittanning (3)
  - Pittsburgh
  - Redstone

West Virginia:

Alma	Kittanning, Lower,	Stockton-Lewiston
Cedar Grove	No. 2 Gas	Winifrede
Chilton	No. 5 Block	Sewell
Coalburgh	Pittsburgh	Fire Creek
Eagle	Powellton	Beckley
Freeport, Upper	Redstone	Pocahontas Nos. 3, 4, 5, 6
Freeport, Lower	Sewickley	Welch
Cedar Grove		

Virginia:

Banner, Upper	Taggart
Banner, Lower	Pocahontas Nos. 3, 5
Imboden	Harlan

Kentucky:

Amburgy	Harlan	Keokee
Blue Gem	Hazard Nos. 4, 6	Millers Creek
Doan	High Splint	Straight Creek
Elkhorn Nos. 1, 2, 3 and 4	Jellico	Wallins
Flag	Kellioka	Whitesburg
Haddix		

Tennessee:

Blue Gem	Mingo
Coal Creek	Red Ash
Dean	Rich Mountain
Jellico	Sterling

Ohio:

Brookville	Pittsburgh
Upper Freeport	Quakertown
Middle Kittanning	Redstone
Lower Kittanning	Sharon

Lake coals were classified into 99 pools according to railroads delivering to Lake Erie ports, districts, use, seams, size (grade).

Ore and Coal Exchange List of Coal Pools

Pool	
1 & 2	Hocking and Pomeroy thin vein ¾-in. lump
3	Hocking and Pomeroy, all mines fresh slack
4	West Virginia splint mine run
5	Hocking thick vein ¾-in. lump
6	Hocking No. 7 vein ¾-in. lump
7	Hocking thick vein mine run
8	Hocking thick vein mine run

Ore and Coal Exchange List of Coal Pools.—(*Continued*)

Pool	
9	West Virginia lump
10	West Virginia splint lump
11	West Virginia splint slack
12	West Virginia slack
13	West Virginia Calumet & Hecla special
14	West Virginia: Gas mine run for steam
15	Gas lump
16	Gas mine run for by-product
17	Gas Kingston mine run
18	Gas slack
19	Gas Mt. Carbon mine run
20	New River-Pocahontas, thick vein lump
21	Pocahontas thick vein nut
22	New River and Pocahontas thick vein mine run
23	Slack, or nut and slack
24	Pocahontas Tug River district, lump
25	Nut
26	Mine run
27	Nut and slack or slack
28	Pocahontas thick vein egg
29	Pocahontas Tug River district egg
30	Elkhorn thick vein mine run
31	Big Sandy and Millers Creek lump
32	All by-products coal, including Harlan & LaFollette mine run
33	All thin vein Kentucky slack
34	Hocking thin vein lump
35	Hocking thin vein slack
36	Jackson thin vein lump
37	All thin vein Kentucky slack
38	Elkhorn thick vein mine run
39	Elkhorn thin vein mine run
40	Pittsburgh: Gas, ¾-in.
40	¾-in.
41	Slack
42	Pittsburgh Steam, ¾-in.
43	Slack
44	¾-in.
45	Greensburg lump
46	Greensburg mine run
47	By-product ¾-in.
48	(Klondike Region) coking coal
49	(Connellsville) mine run
50	No. 8 ¾-in.
51	No. 8 slack
52	Gas by-product mine run
54	No. 8 stripping
55	Hazard Kentucky lump
56	All other Kentucky domestic lump
57	Kentucky steam mine run
58	Kentucky gas mine run

ORE AND COAL EXCHANGE LIST OF COAL POOLS—(*Continued*)

Pool	
60	Amsterdam ¾-in.
61	Amsterdam slack
68	Goshen slack
69	Cambridge mine run
70	Cambridge ¾-in.
71	Cambridge slack
72	Goshen ¾-in.
73	Coshocton ¾-in.
74	Butler-Mercer district slack
75	Butler-Mercer district ¾-in.
76	Freeport ¾-in.
77	Freeport slack
78	Freeport mine run
79	Freeport (by-product) mine run
80	Kenova & Thacker, splint lump
81	Mine run
82	Slack
83	N. & W. Ry. by-products lump
90	Fairmont steam, ¾-in. lump
91	Gas slack
92	Gas ¾-in. lump
93	Other than Pitts seam ¾-in.
94	Mine Run

MISCELLANEOUS LISTS

The United States Fuel Administration (1917–18) is responsible for another classification of coals, as used to fix prices. This classification was based on:

1. Location of mines as to state, district, county, township and location of individual mines.
2. Seams mined.
3. Size of coals: (*a*) run-of-mine, (*b*) prepared sizes, (*c*) slack or screenings.

Coal Classification by Uses

In making a satisfactory commercial classification, the question of use must be carefully considered; its importance is made obvious by the number of uses to which coal may be put and by the fact that use is today the most universally accepted basis for commercial classification. The various uses of coal may be classified as follows:

Commercial Uses of Coal

STEAM-MAKING

Power: Locomotives { Hand-fired. / Stoker fired.

Stationary plants { Hand-fired. / Stoker-fired. / Pulverized coal.

Marine—Bunker coal.

Heating: Central stations { Hand-fired. / Stoker-fired. / Pulverized coal. }

Buildings { Hand-fired. / Stoker-fired. }

Houses.

CARBONIZATION

Beehive coke ovens.
By-product coke ovens.
Gasmaking retorts and generators.

GASIFICATION

Producers.
Water-gas generators.

The British Government coal consumption statistics are given on a somewhat different group use basis, as follows (Report of the Royal Commission, 1925):

Coke ovens (metallurgical coke)
Gas works
Manufactured fuel
Electricity generating stations
Railway companies (for locomotive use)
Vessels engaged in coastwise trade (bunkers)
Blast furnaces (pig-iron manufacture)
Collieries (power)
Miners coal
Domestic Use
General manufacturers and all other purposes
Foreign bunkers

Metallurgical and Manufacturing Processes—Heating, Boiling, Drying, Molting, Freezing (Solid or Pulverized Coal):

Iron and steel manufacture: furnaces and forges; coal solid or pulverized.
Other metals, smelting and refining (including smelting and forging); furnaces.
Cement manufacture: kilns; coal solid or pulverized.
Glass manufacture: furnaces; coal or pulverized.
Brick, tile and pottery manufacture: kilns; coal solid or pulverized.
Paper manufacture.
Textiles (bleaching, dyeing, finishing).
Bread and bakeries.
Refrigeration.
Chemical processes.
Lime.
Petroleum refining.
Rubber manufacture.

Another U. S. Government classification based on use is one that originated with the U. S. Geological Survey, classifying consumption of coals by groups of consuming industries, as follows:

Anthracite

Power and heat at coal mines.
Railroads.
Artificial gas plants.
Electric utilities, including electric railways.
Domestic trade (domestic sizes not included elsewhere).
Steam trade (steam sizes not included elsewhere for heating and industrial use).

Bituminous

Power and heat at coal mines.
Mines and quarries, other than coal.
Railroads.
Public utilities:
  Electric, including electric railways.
  Gas.
Beehive coke plants.
By-product coke plants.
Bunkers:
  Foreign.
  Coastwise and Lake trade.
Iron and steel works, not including coal for coke.
General industrial and other unspecified uses.
Domestic consumers.

Commercial Classifications of Foreign Coals

Great Britain

Based on kind (bituminous and anthracite), size, production district and shipping port, seam, use, structure, quality.

For market purposes, the classifications are as follows:

South Wales

Steam	Bituminous	Anthracite
Best Admiralty large	No. 3 Rhonda large	Best large
Bests Seconds	No. 3 Rhonda smalls	Seconds large
Drys—Best	No. 2 Rhonda large	Red Vein large
Dry—Ordinary	No. 2 Rhonda through	Machine made cobbles
Best Black Vein large	No. 2 Rhonda smalls	French nuts
Western Valleys		Stove nuts
Eastern Valley		Machine beans
Steam smalls		Machine peas
Coking smalls		Rubbly calm
Washed duff		

Swansea District

Steam	Bituminous
Best large	Large
Best through	Through
Best smalls	Smalls

Northumberland

Best Blyth screened, steam and house	Tyne second steams
Best Blyth smalls	North North'd. steams
Blyth second steams	North North'd. smalls
Best Tyne screened steam and house	North'd. unscreened (for bunk)
Best Tyne smalls	Northumberland nuts
Tyne second smalls	Northumberland smithies

Notts, Derby and Yorkshire

Best South Yorkshire hards
West Yorkshire Hartleys
Yorkshire screened steam
Trebles, washed
Doubles, washed
Dry nuts
Singles, washed
Smalls, washed
Rough slack
Nutty slack
Best Derbyshire large
Screened steam, f.a.s.
Grimsbyor Lummingham

Durham

Best screened steam
Second screened steam
Durham gas coal (wear special)
Durham gas coal (Tyne second)
Durham gas coal (Tyne prime)
Durham bunkers, best
Durham bunkers, ordinary
Coking coal, unscreened and smalls
Peas and nuts

Lancashire, Yorkshire, and Derbyshire

Best Lancashire steam screened
Number unscreened
Best South Yorkshire, hards
Best Derbyshire hards
Slack, dry

Scotland

West of Scotland

Ell coal (best)
Best splint
Second splint
Best navigation screened
Best steam screened
Best Hartley

Fifeshire

Best navigation screened
Second navigation screened
Best navigation unscreened
Second navigation unscreened
First navigation smalls
First class steam
Third class steam
Treble nuts, washed
Double nuts, washed
Singles and beans
Pearls

Lothian

First Hartley
Second Hartley
Trebles
Doubles
Singles
Pearls

Belgium

Industrial Coals

	Approx. Analysis Vol., Per Cent.	Ash, Per Cent
Coking coal:		
Washed smalls, 0/8	18/24	8/10
Unscreened 25 per cent	18/24	12/16
35 per cent	18/24	12/16
Nuts 50/80 mm	18/24	10/12
30/50 mm	18/24	10/12
15/30 mm	18/24	10/12
10/15 mm	18/24	10/12
Screened	18/24	10/12
Gas Coals:		
Washed smalls 0/8	28/35	8/10
Unscreened 25 per cent	28/35	12/16
35 per cent	28/35	12/16
gas 25 per cent	28/35	12/16
gas 35 per cent	28/35	12/16
Nuts 50/80 mm	28/35	10/12
30/50 mm	28/35	10/12
15/30 mm	28/35	10/12
10/15 mm	28/35	10/12
Screened	28/35	10/12
Bituminous:		
Unscreened 25 per cent	13/15	12/16
Cobbles 50/80 mm	13/15	10/12
Nuts 30/50 mm	13/15	10/12
Beans 15/30 mm	13/15	10/12
10/15 mm	13/15	10/12
Screened	13/15	10/12
Blast-furnace coke	13/15	10/12

Germany

1. Anthracite
2. Pitcoal (bituminous)
3. Brown coal (lignite)

Market descriptions.

1. By districts and mines.
2. By sizes
    - Run of mine, with different percentages of lump.
    - Lump, large
    - Nut I, II, III, IV.
    - Mixed
    - Smalls
3. By use
    - Coking
    - Gas
4. By flame character
    - Flaming or fat coal (low volatile)
    - Gas-flame coal (high-volatile)
5. By preparation
    - Washed (by sizes)
    - Unwashed

France

Market classifications of coals produced in France; these do not include indemnity coals or imported coals.

Industrial Coals

Raw Coals: (Flaming, vol. mat., 32% and over; bituminous, vol. mat., 20% and over)

Duffs, 0/10
0/20
0/30
0/50
Unscreened, 20/25
30/35
Dry peas, 20/40

Washed Coals (Flaming, vol. mat., 32% and over; bituminous, vol. mat., 20% and over)

Duffs, 0/10
0/20
Peas, 10/20
10/30
Smithy, 10/30

Domestic Coals

Screened, 10
20
40
80
Lumps and gaillets
Unwashed cobbles, 40/80
20/80
Washed peas, 20/40
nuts, 30/50
beans, 20/30

Raw coals (Semibit., vol. mat., 13 to 20%; quarter-bit., vol. mat., 11 to 13%; lean, vol. mat., 11 per cent. and under)

Duffs, 0/10
0/30
0/50
Unscreened 20/25
30/35
Dry peas, 60/30

Washed Coals (Semibit., vol. mat., 13 to 20%; quarter-bit., vol. mat., 11 to 13%; lean, vol. mat., 11% and under)

Duffs, 0/10
0/30

Washed Coals (With 1/3 peas, semibit., vol. mat. 12 to 20%; quarter-bit., vol. mat., 11 to 13%; lean, vol. mat., 11% and under).

Duffs, 0/30
0/30 (Semiwashed 15% ash)
Peas 10/30
7/20
5/15
5/10

India and South Africa

Mention should also be made of the system of grading coals for export put into effect recently in India and South Africa, by Coal Grading Boards, established by producers and consumers, with the active cooperation of the local governments.

The grading in each country applies on coals for export and mines are classified as well as coals.

India

The grades fixed by the Indian Board are as follows:

	Low-volatile	High-volatile
Selected grade:	Up to 13 per cent. ash and over 7000 cal., or 12,600 B.t.u.	Up to 11 per cent. ash; over 6800 cal. or 12,240 B.t.u. and under 6 per cent. moisture.
Grade No. I:	Up to 15 per cent. ash and over 6500 cal. or 11,700 B.t.u.	Up to 13 per cent. ash; over 6300 cal. or 11,340 B.t.u. and under 9 per cent. moisture.
Grade No. II:	Up to 18 per cent. ash and over 6000 cal. or 10,800 B.t.u.	Up to 16 per cent. ash; over 6000 cal. or 10,800 B.t.u. and under 10 per cent. moisture.
Grade No. III:	All coals inferior to above.	

South Africa

The Coal Act No. 27, which became operative Sept. 1, 1922, provided for:

1. Compulsory grading of coal intended for export or for bunkering.
2. Appointment of a committee to control grading.
3. A monetary levy upon the various collieries, based upon the tonnage shipped, to defer the expense of grading.

The classification adopted was somewhat similar to that used in India, but full details are not at present available. They will, however, be secured as soon as possible.

Conclusions

It is obvious that the establishment of a commercial classification of coals that will be simple, comprehensive and accurate will be a difficult matter; it is equally obvious that such a classification is needed today.

As to the proper bases to be used: in all classifications now in use, four characteristics are apparently most generally included; *i. e.*, use, locality and seam, percentage of volatile matter, rank or class. To combine these bases in an accurate, clear, nontechnical and brief classification system would seem to be at last one method of attacking the whole problem and might lead to its satisfactory solution.

DISCUSSION

N. G. Alford, Pittsburgh, Pa., on inquiring as to whether the paper was offered as a recommendation for adoption was advised that it was not.

Commercial Description of Pennsylvania Anthracite

By E. W. Parker,* Philadelphia, Pa.

(New York Meeting, February, 1930)

Anthracite, as sent to market, comes under three general terms of description: characteristics, source and size.

Anthracite is generally classified as white ash, red ash, or Lykens Valley. The white ash coals are sometimes further described as "hard" or "free-burning." Lehigh white ash would come under the former description and Shamokin white ash under the latter. As a rule, it may be said that the nearer the origin of the coal to the western extremity of the anthracite region, the softer and more free-burning it is. Indeed, for many years the coal from Trevorton, at the western tip of the Middle Western field, was classified as "semianthracite" in the shipment reports.

Lykens Valley coal is always a red ash coal, with a higher volatile content than the general run of anthracite, and consequently free-burning. The name does not exclusively indicate the neighborhood in which the coal is dug, but rather refers to the beds from which it is obtained. These beds lie, not above the Pottsville conglomerate, as is the rule, but encased in it. The extreme eastern limit of the Lykens Valley beds would seem to be about Tremont, in Schuylkill County, and the Natalie mine, near Mount Carmel. There are, of course, traces of these beds farther to the east, but they do not seem to be workable. There are as many as six of the Lykens Valley measures found in the conglomerate at some points, but to a large extent they are unworkable. Whether this coal is obtained in the Lykens Valley proper, where the beds were first identified, or from the Shamokin Valley, it is always Lykens Valley coal, and it has long been considered a premium coal, especially desirable for domestic use, particularly when a quick, hot kitchen fire is desired.

Coals are, of course, frequently marketed under special names—Jeddo, Scranton, Kingston, Old Company Lehigh, Famous Reading, Lackawanna, etc.—which, of course, have the trade significance which their producers and vendors have built up.

General Description of the Various Coals, with Name of Seam and Producing District

This is partly covered in the paragraphs above. Lykens Valley coal is produced from one or another of the six Lykens Valley beds lying within

* Director, Anthracite Bureau of Information.

the Pottsville conglomerate. It is always red ash, it has a rather high volatile content, and is always free-burning. It is mined chiefly in the Lykens Valley of Dauphin County and in the west end of Schuylkill County, principally by the Susquehanna Collieries Co. and the Philadelphia & Reading Coal & Iron Co. It is also produced in the Shamokin Valley, notably at the Cameron and Luke Fidler mines of the Susquehanna Collieries Co. and at the Natalie mine of the Colonial Collieries Co.

White ash coal is produced in all parts of the anthracite region. It is supplied by every characteristic bed beginning with the C, Gamma or Five-foot (the third bed counting up from the conglomerate) up to and including the Holmes. This, of course, embraces the Mammoth bed, which itself supplies a good percentage of the total anthracite mined. The extent to which white ash coal predominates may be gathered from the following figures, as reported by Daddow & Bannan:

At Pottsville (where the red ash beds are most abundant), out of 113 ft. total thickness 49 are in pure white ash beds, with 10 ft. more in a so-called "gray ash" or "pink ash" bed.

On Broad Mountain, in a total thickness of 100 ft., 75 ft. are white ash.

At Beaver Meadow, the entire thickness of 45 ft. is white ash.

At Mahanoy City, out of 84 ft. total thickness, 58 ft. are in white ash.

In general, it might be said that the anthracite beds in the Pottsville basin can be listed thus:

1. Little Buck Mountain. This is the first bed above the Pottsville conglomerate. Red ash. Next, in the ascending order is:
2. Buck Mountain. Red ash, at least in the lower bench.
3. Five-foot. White ash.
4. Skidmore, known as Wharton in the Lehigh region. White ash.
5. Mammoth. The predominant bed of the anthracite region. White ash.
6. Seven-foot, by some regarded as a mere split or leader of the Mammoth. White ash.
7. Holmes. This is the last true white ash bed in the ascending order.
8. Primrose. This bed is variously known as a gray ash or a pink ash, according to situation.
9. Orchard. Red ash.
10. Little Orchard. Red ash.
11. Diamond. Red ash.
12. Tracy. Red ash.
13. Little Tracy. Red ash.
14. Gate. Red ash.
15. Sandrock. Red ash.

For other districts, Daddow & Bannan (Coal, Iron and Oil, pp. 169 et seq.) give these figures in their various columner sections, and in ascending order:

At Carbondale: Five-foot bed, 4 ft.; Skidmore, 5 ft.; Mammoth and Seven-foot, 24 ft. combined; Holmes, 6 ft. All white ash.

At Scranton: Little Buck bed, 2 ft., red ash; Buck Mountain, 5 ft., red ash; Five-foot, 6 ft., white ash; Skidmore, 8 ft., white ash; Mammoth and Seven-foot, 14 ft., white ash; Holmes, 6 ft.; Primrose, 12 ft.; Orchard, 7 ft.; Little Orchard, 5 ft. It is not stated whether the upper beds are red ash in that field.

The same authority quotes beds and thicknesses in other neighborhoods as shown in Table 1.

Table 1.—*Thickness, in Feet, of Various Beds*

District	Little Buck	Buck Mountain	Five-foot	Skidmore	Mammoth & Seven-foot	Holmes	Primrose	Orchard	Little Orchard	Diamond	Tracy	Little Tracy	Gate	Sandrock
Wyoming Basin	5	?	?	7	20	7	5	7	7	7	8			
Beaver Meadow			7	8	30									
Hazleton	?	12	6	8	30									
Harleigh		11	10	12	30									
Mahanoy City	8	8	16	10	25	7	12	8						
Locustdale	6	16	6	10	25	7	12	8	6					
Shamokin	5	12	4	4	18	?	8	8	6					
Tamaqua	5	8	15	8	23	3	12	5	3	6				
Broad Mountain	7	18	6	9	60									
Pottsville	3	10	5	8	32	4	10	6	3	7	9	4	9	3

In the Pottsville basin there are two small beds between the Little Orchard and the Diamond; two more between the Little Diamond and the Tracy; and two more between the Little Tracy and the Gate. The "E" bed of Daddow & Bannan (Mammoth and Seven-foot) is known as the Baltimore bed in the Wyoming Valley. The Gate vein, peculiar to the Pottsville basin, has been variously known as the Salem, Spohn, Peach Mountain and Lewis. It produced a coal especially favored for domestic purposes in Philadelphia a century ago.

Dr. H. H. Stoek made a summary of the characteristics of the different anthracite fields, which are summarized here:

Northern, or Wyoming-Lackawanna: Impossible to give the whole number of workable beds, but 81.8 per cent. of total coal in this field is marketable.

Eastern Middle, or Lehigh: 75 to 77 per cent. of total thickness marketable.

Western Middle, or Mahanoy-Shamokin: Total coal measures 1200 ft. thick, with 10 to 12 beds. Lykens Valley measures found in the conglomerate. About 75 per cent. of total thickness is marketable.

Southern, or Schuylkill: Coal measures 2500 ft. thick, and six Lykens Valley beds are present in the conglomerate in the west end; 20 beds have been worked and 72 per cent. of the coal is marketable.

Dr. Stoek's figures for total thickness and thickness marketable—which is not the same thing as recoverable—at various points are given in Table 2.

TABLE 2.—*Total and Marketable Thicknesses*

District	Total Thickness, Feet	Amount Marketable, Feet
Pottsville	108	78
Tamaqua	109	78
Shamokin	70	54
Shenandoah	113	87
Wilkes-Barre	85	69

PHYSICAL CHARACTERISTICS

It is impossible to tie down the physical characteristics of anthracite, at least with respect to appearance. Its fractures range from flaky and shelly up to nearly cubical. Conchoidal fracture predominates. Its color ranges from a clear, shiny black, into bluish and grayish tinges, with or without luster. These characteristics not only vary from place to place, but they vary as between two beds in one mine, or even as between two points in the same bed. It has been a sort of mining tradition that the cubical and conchoidal fracture was characteristic of coal from the Wyoming-Lackawanna field, while the flat was more abundant in the Southern field. In the same way the jet color has been held characteristic of Wyoming coal, and the blue or gray tinge of Southern field coal. Yet, the Diamond bed in the Pottsville basin is hard, pure and shiny, with conchoidal fracture, and in the Little Tracy the coal is good and clean, with few faults or impurities, while in the intermediate Tracy bed the coal is soft and shelly.

SOURCES

In this respect, anthracite is usually spoken of as "fresh-mined," "washery," and "dredge" or "river" coal. The names are virtually self-explanatory. Fresh-mined coal is that which is mined, prepared and shipped in a continuous process. Washery coal is that reclaimed from culm banks. If it was good coal in the first place, and the washery preparation is up to par, washery coal is a good fuel, since anthracite does not deteriorate upon exposure. Dredge coal is fine material recovered from creeks and rivers fed by water from the coal region,

which carries down more or less solid material from dirt banks. This reclaimed coal, if really clean, gives good results, but the difficulties in removing sand, ash, and other impurities scooped up by the dredges sometimes prove almost insuperable, to the detriment of the fuel value of the product.

Sizes

Anthracite is finally classified commercially with respect to sizes. The standard testing screens for domestic coal are given below, all meshes being round:

	Through, Inches	Over, Inches
Broken	4 7/16	3 7/16
Egg	3 7/16	2 8/16
Stove	2 8/16	1 9/16
Chestnut	1 9/16	11/16
Pea	12/16	8/16

In breaker operation some smaller coal will necessarily be carried over the screens. Standard preparation permits of not more than 15 per cent. of undersize in Broken, Egg and Stove coal, and not more than 10 per cent. in Chestnut and Pea coal. Standard preparation also provides that Egg coal should not contain more than 3 per cent. of bone and 2 per cent. of slate, Stove coal not to exceed 4 per cent. of bone and 3 per cent. of slate, and Chestnut not to exceed 5 and 4 per cent. respectively.

Below these "domestic sizes" there are, in the descending scale, Buckwheat No. 1, Buckwheat No. 2 (Rice), Buckwheat No. 3 (Barley) and down to silt. Mixtures of smaller sizes are sometimes classified as "boiler coal." In addition, lump coal is still made, though in decreasing quantities and usually only on special order. This is coal in lumps about the size of a man's head, and it is in such small demand that in 1924 only 13,209 tons were shipped and in 1926 the total was only 5187 tons. In 1928 the production of lump coal was too small to report separately. When anthracite was a blast-furnace fuel this was one of the standard sizes, running into seven figures in total shipments of 30,000,000 tons and even less (1,273,104 net tons in 1864, for example).

Use of Anthracite

The bulk of anthracite production, of course, goes into domestic consumption, and aside from any distinct trade which may have been built up by the use of a trade name very little has been done—or prob-

ably can be done—to develop a call for this or that coal. Production is usually segregated into white ash and red ash, and further divided into hard and free-burning, as described. Different markets may have different tastes, which dealers keep in mind when buying. But anyone who can handle an anthracite fire actually has but little choice between white ash and red, or between a hard anthracite and a moderately free-burning one. It can readily be seen, too, that where the districts have anywhere from 4 to 14 beds there would be little incentive to build up a trade for one particular bed of coal. Besides, in the lower part of the anthracite region, at least, it is a sort of a maxim that no operation gives any promise of profit unless the Mammoth is available. To a large degree, therefore, anthracite is anthracite, just as "pigs is pigs." Several beds are sometimes worked from the same shaft at the same time, the entire product going through the breaker without discrimination.

With respect to sizes, it might be said that anthracite can be further classified with regard to the uses for which it is best adapted, thus:

Broken: For large hot-air furnaces, as in churches and halls; gas manufacture.

Egg: The best all-around fuel for domestic hot-air furnaces; gas manufacture; brass foundries.

Stove: The standard domestic fuel, especially for heating plants, whether hot air, steam, or hot water.

Chestnut: Domestic fuel, serviceable in heating plants of all descriptions, and especially adapted to kitchen ranges and small stoves.

Pea: Economical fuel for laundry stoves, small heaters for hot-water tanks, and especially useful for mixing with Chestnut coal when it is desired to hold a fire without forcing.

The smaller sizes, from Buckwheat No. 1 down, are known collectively as "steam" sizes, although Buckwheat No. 1 has proved its value as a domestic fuel when used in steam or hot-water heating equipment fitted with magazine feeds or with mechanical stokers. The smaller sizes are suitable for the boiler rooms of factories, either by themselves or mixed with bituminous coal, and are especially desirable for low-pressure heating systems, as in hotels and apartments where safe, clean and efficient fuel is demanded.

Approximate Tonnages Produced of Each Size of Anthracite

The bulk of the tonnage is, of course, white ash. What the proportions are is hard to say. As to proportions of sizes, the answer is easy. The total shipments in 1928, according to the United States Bureau of Mines, were 59,364,00 tons, as compared with 67,249,000 tons in 1926, and 68,971,000 tons in 1924. The quantities and percentages of the different sizes in these years were as shown in Table 3.

TABLE 3.—*Production of Anthracite*

	1928		1926		1924	
	Shipments, Tons	Size, Per Cent.	Shipments, Tons	Size, Per Cent.	Shipments, Tons	Size, Per Cent.
Broken	382,000	0.64	1,060,000	1.58	1,844,000	2.67
Egg	6,749,000	11.37	8,923,000	13.27	9,843,000	14.27
Stove	14,778,000	24.90	16,616,000	24.71	15,038,000	21.80
Chestnut	14,761,000	24.87	18,002,000	26.77	18,100,000	26.24
Pea	4,597,000	7.74	3,699,000	5.50	5,689,000	8.25
Total domestic	41,267,000	69.52	48,300,000	71.82	50,514,000	73.24
Buckwheat No. 1	7,830,000	13.19	7,663,000	11.39	8,491,000	12.31
Buckwheat No. 2	4,800,000	8.09	5,368,000	7.98	4,585,000	6.65
Buckwheat No. 3	4,685,000	7.89	4,683,000	6.96	4,642,000	6.73
Boiler	85,000	0.14	354,000	0.53	107,000	0.16
All other	697,000	1.17	881,000	1.31	632,000	0.92
Total steam	18,097,000	30.48	18,949,000	28.18	18,457,000	26.76
Grand total	59,364,000	100.00	67,249,000	100.00	68,971,000	100.00

DISTRIBUTION OF ANTHRACITE

Based upon reports for the coal year 1923–24, shipments of anthracite within the United States were 74.8 per cent. domestic sizes (Pea and larger) and 25.2 per cent. steam sizes. Of the coal so distributed, the principal amounts went thus:

	Pea and Larger, Per Cent.	Steam Sizes, Per Cent.
New England	16.8	2.1
New York, New Jersey and Pennsylvania	64.9	94.8
Maryland, Delaware, District of Columbia and Virginia	4.3	0.5
Ohio, Indiana, Michigan, Wisconsin, Minnesota	12.5	2.3
	98.5	99.7

Properties of Coal Which Affect Its Use for the Manufacture of Coal Gas, Water Gas and Producer Gas

By Gilbert Francklyn,* New York, N. Y.

(New York Meeting, February, 1930)

The requirements of coals for the manufacture of coal gas, water gas and producer gas will be considered separately and a short description of each gasmaking process will be given.

In this report the term "gas coal" is confined to coals suitable for the manufacture of city gas in horizontal, inclined or vertical retorts. The report does not refer to coke-oven practice.[1]

Briefly, the process of manufacture consists of distilling the coal at high temperature in an externally heated retort out of contact with the air. The products of distillation, consisting of gas, tar, ammonia, etc. are collected and purified. The residue consists of coke, which is removed from the retort at the end of the carbonizing period. To supply the heat required in the process, a portion of the coke produced is converted into producer gas, which is burned under the retort settings.

Qualities of Gas Coals

The chief qualities that coal must possess in order to be classified as a gas coal are as follows:

1. *Gas Yield.*—The coal must be capable of yielding gas in sufficient volume and of a satisfactory quality. The yield depends considerably upon the temperature of distillation; a higher temperature will give a larger yield of gas, but the quality of gas (now measured by the British thermal units per cubic foot) will be lower, while a lower carbonizing temperature will give less gas of higher heating value. The true measure of the yield is the number of British thermal units produced in the gas from a given quantity of coal. This number is obtained by multiplying the cubic feet of gas from 1 lb. of coal by the British thermal units per cubic foot of gas; it is called B.t.u. feet per pound. It is found that the

* Fuel Agent, Consolidated Gas Co. of New York and Member of Use Classification Technical Committee of Sectional Committee on Classification of Coal. American Standards Association.

[1] For this information, see H. J. Rose: Selection of Coals for the Manufacture of Coke. *Trans.* A. I. M. E. (1926) **74,** 600.

B.t.u. feet per pound are not much affected by ordinary changes in the temperature used for carbonization. In practical work, a gas coal should be capable of yielding 3000 B.t.u. ft. per pound, when carbonized in a well-operated plant with retorts in good repair. This yield should, of course, be obtained without steaming and without any mixture of furnace gas through the retort walls. The only laboratory indication as to the probable yield of a coal is the percentage of volatile matter in the proximate analysis, and this affords only a rough basis of comparison of the yield of different coals. It is unfortunate that there is, at present, no standardized laboratory method in widespread use, for testing coals for their yield of gas in B.t.u. feet per pound. Such a test of a gas coal would be somewhat analogous to the calorific test now used on steam coal.

2. *Coking Qualities.*—The coal must produce a coke suitable to the uses to which that coke is to be put. One of these uses will always be the production of heat for the process of carbonization. The coke must, therefore, be a suitable fuel for the bench producers. The other uses of the coke will vary with local conditions. The qualities generally required in coke are good size, good structure, low ash and high or moderately high fusibility of ash, absence of tendency to form clinker, and low sulfur. A high yield of coke need not be required because it will generally be obtained at the expense of the yield of gas.

3. *By-products.*—In addition to gas and coke, the process of carbonization will yield certain quantities of tar and ammonia, the sale of which is an important credit item against the cost of the coal. There is a difference in the yield of tar and ammonia from different coals, but the difference is not great enough to be considered a controlling factor in the suitability of the coal for most gas plants. The yield of other by-products, such as cyanogen and naphthalene, and light oils, need not generally be considered.

4. *Ash, Moisture and Sulfur.*—Low ash is an important consideration because the ash displaces valuable ingredients. All the ash in the coal remains in the coke, and high-ash coal therefore will produce a coke of still higher ash content. A gas coal should have as little ash as possible and should generally analyze below 8 per cent.

Moisture, like ash, displaces valuable ingredients and is also objectionable because additional heat must be applied to vaporize it during carbonization. Most of the gas coals available do not contain excessive moisture.

Sulfur is an objectionable impurity. A portion of the sulfur appears in the gas and a portion remains in the coke; the quantity in each case depending upon the original form of the sulfur in the coal. A small amount of the total sulfur in the coal is also found in the tar produced. The sulfur in the gas, which is in the form of hydrogen sulfide, is removed by purification. An increase in its quantity increases the cost of purifica-

tion. The sulfur remaining in the gas after purification is in the form of so-called organic sulfur compounds. These form sulfur dioxide when the gas is burned. Sulfur remaining in the coke may be objectionable for certain uses and is often associated with a tendency to form excessive clinker. For these reasons, it is usual to insist that the sulfur content of gas coals shall be less than 1.25 per cent.

5. *Physical Qualities.*—The fusion point of the ash should not be below 2350° or 2400° F., when measured by the A. S. T. M. standard method, because a low-fusing ash is apt to cause clinker when the coke is burned. This point is important in a coal with high percentage of ash. When the percentage of ash is low, there is less likelihood of clinker trouble. The structure of the coal is not of great importance unless it is so friable as to cause an excessive quantity of fines, which may prevent even charging of the retorts. If the coal is to be stored for any length of time, lumpy coal will generally show less deterioration.

Sizing of the coal is not necessary, but screening out the fines is usual in coals from certain mines. The chief reason for this is that the fines from these mines are found to contain more impurity than the lumps, and the screened coal therefore shows a better analysis.

Summary of Qualifications

Summarizing the qualifications listed above, we may say that a gas coal must be a coking coal and will generally show a proximate analysis somewhat as follows:

	Per Cent.	
Moisture	2	High limit about 4 per cent.
Volatile matter	35	Low limit about 32 per cent.
Fixed carbon	57	
Ash	6	High limit about 8 per cent.
	100	
Sulfur	0.9	High limit about 1.25 per cent.
Fusion point	2500° F.	Low limit about 2350° F.

It will usually have a fuel ratio (fixed carbon divided by volatile matter) between 1.50 to 2.00.

Such a coal would be described under Dr. Ashley's classification as a bituminous caking coal "Hivol," or Coal 56 in rank, and the grade would be described as A.S.(or T.)G.

It should be realized that none of the widely used laboratory tests to which coals are now submitted will show positively the suitability of a coal for gasmaking. It may be said, however, that a coal which shows an analysis within the limits given above, and complies with the other requirements mentioned, will in all probability be a satisfactory gas coal.

Coal for Water-gas Generators

For the purpose of this report, coal for water-gas generators will be referred to as generator fuel. The process to which this coal is subjected is intermittent. The generator consists of a cylindrical chamber lined with refractory material with grate bars near the bottom to hold the fuel bed. This fuel bed is brought to incandescence by combustion with an air blast. The combustion products escape to the atmosphere, after part of their sensible heat has been utilized. When the temperature is high enough the air blast is shut off, the stack valve closed and steam is passed through the fuel bed. This steam is decomposed, uniting with the carbon of the coal to form "blue gas," which is a mixture of hydrogen, carbon monoxide and a small quantity of carbon dioxide. The so-called run continues until the temperature of the fuel bed has dropped too low to give good results, then the steam is shut off, the stack valve opened, and the air blast again turned on. During the run the blue gas is usually carburetted with oil gas, produced by cracking gas oil in adjoining equipment. After carburetting, the gas is cooled and purified for use. In some cases the blue gas may be used without further enrichment. Both anthracite and bituminous coals may be used for this purpose.

Anthracite for Generator Fuel

For many years, anthracite was (with the important exception of coke, which is not considered in this report) the only fuel used for this process. The chemical qualities of most anthracites seem to be very suitable, except that some of them run too high in ash, and that some foreign anthracites are too high in sulfur. Ash is apt to form troublesome clinker and should be as low as possible, certainly not more than 12 per cent. High sulfur requires more purifying capacity than is provided at most plants, and should not exceed 1 per cent. Moisture is undesirable, but is not generally excessive in anthracites.

The physical properties of generator fuel are important. Fusing point of ash should be high; in anthracites from this country it usually exceeds 2900° F. The coal must be sized and must be hard enough to be delivered at point of consumption without breaking into fines. It must not show a tendency to break up into small pieces or into dust when subjected to high temperature. The sizes generally preferred are Broken or Egg. Stove coal can be used, but is usually higher in price. Smaller sizes do not give satisfactory results.

Bituminous Coal Generator Fuel

During the past few years, there has been a tendency to use bituminous coal as generator fuel. It can be used alone or mixed in various proportions with anthracite or coke. The coals used have mostly been

gas coals or coals closely related to them. The physical properties of the coals, however, seem to be more important than the chemical properties. These coals must be sized to lumps of about 3 by 6 in. or 2 by 4 in. (operators have different opinions on this subject) and they must be hard enough to maintain these sizes and to reach the point of consumption free from fines. The fusing point of the ash should be high (2600° F. or higher) to avoid tendency to clinker. As to chemical qualities, the ash should be as low as possible and the sulfur not over 1.25 per cent., for reasons given in the preceding paragraphs. High-volatile coking coals have generally been chosen for this use, probably for the reason that they are better adapted physically than many of the low-volatile coals, which are often too friable. Volatile matter content and coking properties do not appear to be the most important consideration in selecting coal to be used for water-gas manufacture.

Since coals as different in rank as anthracite (noncoking and with more than 90 per cent. fixed carbon on an ash-free basis) and high-rank bituminous coal (strongly coking, with a percentage of fixed carbon as low as 60 per cent.) can both be used with success, it would seem impossible to say definitely what the chemical requirements are for this process, except that in all cases ash, sulfur and moisture should be as low as can be obtained, and coals that show the least tendency to clinker should be chosen. The physical requirements are more definite; they consist of high fusing point of ash (to avoid clinkering troubles) and a structure that permits of proper sizing and will not break up under the influence of high temperature.

Coal for Manufacture of Producer Gas

The process of manufacture of producer gas is continuous. A current of air and steam is passed through a fuel bed contained in a cylindrical producer lined with refractory material. The upper layer of the fuel bed consists of raw fuel, which is fed in from the top and is gradually heated as it passes downwards, with distillation of the volatile components. The middle layer of the bed is mostly carbon and ash, and this layer is kept at a temperature high enough to decompose steam, the temperature being regulated by changing the proportion of air and steam passing through. In this layer the air and steam combine with the carbon to form carbon monoxide, hydrogen and a small quantity of carbon dioxide. These gases, with nitrogen, pass upward, mixing in the upper layer with the products of distillation, and are led away from the top of the producer for use. The lower layer of the bed is composed mostly of ash, which is eventually removed from the bottom of the producer. The quality of the gas produced will vary with the type of fuel and the proportion of air and steam used.

It is essential that the percentage of fixed carbon in the coal shall be sufficient to provide enough heat in the process to drive off moisture and the volatile matter from the upper portion of the fuel bed. Unless this condition is met, there will be difficulty in holding the necessary temperature. It has been stated that the moisture in the coal must not exceed 25 per cent. and that the volatile matter must be less than the fixed carbon.[2] The chief operating difficulty is to keep the fuel bed in such condition that the air and steam pass through it evenly and so make thorough contact with the combustible matter of the coal. High ash, low fusion point of ash and strong coking quality tend to make an uneven fuel bed by the formation of blow holes, and are therefore objectionable. High sulfur is likely to be associated with low fusion point of ash and is also objectionable in the producer gas.

Coals of different sizes can be used but a great variation in size, such as run of mine coal varying from large lumps to fines, makes it more difficult to maintain an even fuel bed.

With the limitations mentioned, coals of all rank, from anthracite to peat (as well as coke), can be used in suitable producers, and the price for which they can be obtained is generally the most important consideration.

Allowable Sulfur in Manufacture of Gas

As bearing upon the chemical qualities of coals suitable for manufacture of gas, the tabulation of requirements as to sulfur (Table 1) adopted by state public utility commissions is submitted. This has been checked from the original data in possession of the American Gas Assn., July, 1928.

---

[2] Keystone Coal Buyers Catalogue, 1928.

TABLE 1.—*Allowable Sulfur in Manufacture of Gas*

State	Hydrogen Sulfide, Maximum Allowable	Sulfur Grains per 100 Cu. Ft. Maximum Allowable
Alabama[a]	trace	30
Arizona	trace	30
Arkansas[b]		
California	trace	30
Colorado[c]	trace	30
Connecticut	trace	30
Delaware[d]		
Dist. of Col.	not to show presence	30
Florida[b]		
Georgia[a]		
Idaho[a]		
Illinois	1 gr. per 100 cu. ft.	30
Indiana[c]	trace	30
Iowa[b]		
Kansas[a]		
Kentucky[a]		
Louisiana[a]		
Maine	trace	30
Maryland[c]	trace	30
Massachusetts	must be absent	30
Michigan[a]	trace	
Minnesota[b]		
Mississippi[b]		
Missouri[c]	trace	30
Montana	trace	30
Nebraska[b]		
Nevada	trace	30
New Hampshire	trace	30
New Jersey	trace	30
New Mexico[b]		
New York[c]	trace	30
North Carolina	trace	30
North Dakota	trace	30
Ohio[a]		
Oklahoma[a]		
Oregon	trace	25
Pennsylvania		30
Rhode Island[a]		
South Carolina	trace	30
South Dakota[b]		
Tennessee[a]		
Texas[a]		
Utah[a]		
Vermont[a]		
Virginia[a]		
Washington[c]	trace	30
West Virginia[a]		
Wisconsin	trace	30
Wyoming[a]		

[a] Commission has no general regulations.
[b] Commission has no jurisdiction.
[c] State rules do not apply in some cities.
[d] No commission.

DISCUSSION

G. H. ASHLEY, Harrisburg, Pa., asked if any progress is being made in the elimination of sulfur, stating that in a furnace burning gas with an appreciable sulfur content, it is necessary to use metal costing about 35 c. per lb., to prevent corrosion, whereas, if the gas were free from sulfur metal costing only a tenth as much could be used.

MR. FRANCKLYN replied that while H_2S is now almost entirely removed, the organic sulfur remains in the gas.

H. J. ROSE, Pittsburgh, Pa., verified this, adding that organic sulfur is ordinarily removed at gas plants only when it is excessively high, a common limit being 30 gr. per 100 cu. ft. Such sulfur may be reduced by oil washing, although processes are now available whereby it can be entirely removed.

GENERAL DISCUSSION OF FEBRUARY, 1930, COAL CLASSIFICATION PAPERS

T. F. Downing, Jr., Philadelphia, Pa. (written discussion).—The papers presented at the February, 1930, meeting acquainted us with the tremendous amount of study and labor which the committees have already given to the subject. We also got some idea of the work still remaining to complete the job.

The object of this paper is to present some observations of the writer concerning points not previously discussed but which would seem pertinent to a complete classification.

In examining a virgin property the engineer takes face or drill core samples for analyses. From the results he roughly classifies the coal and computes the return which may be expected from sales in the markets tributary to the field. Incomplete consideration of this part may result in serious loss to his client or company.

When called in on marketing problems the engineer should first visit the mines, take samples, observe mining conditions, and generally compare the value of the coal with those of other producers in the same trade.

The large consumer should likewise be somewhat familiar with conditions at mines from which his supply comes so as to be better able to judge value received.

The writer has been fortunate in having to look on the subject from all sides and the following expressions are the result of experience.

It will be noted that the commercial value of coal is expressed in dollars and cents whether the viewpoint be that of the owner, producer, or consumer. It would therefore seem that the commercial success of the proposed classification will depend on how close it shows the relative values of our American coals.

Analyses of bed sections do not always reflect the possibilities or value of the seam. The writer has never read any discussion on this particular phase but is firmly convinced that operating methods and screening may change the chemical qualities of the product as shipped.

Mining men use the term "mining section" to designate a part of a seam softer than other parts and in which the coal can be most easily undermined or cut. Many veins have this soft stratum and some beds are made up of several layers of coal each differing in physical structure.

When a seam contains strata of unsimilar physical characteristics the chemical analysis of each stratum usually differs from those of the others. This is a fact not generally recognized. The writer is familiar with a considerable area whose coal would be rejected for cokemaking, on analysis of face samples, because of high sulfur content yet the nut and slack contain less sulfur than the standard set up in the papers presented at the meeting.

To illustrate better, and form a definite basis for partial discussion on the above points, actual analyses from a single face sample of a well known Eastern bed are given here. The information must be treated as confidential so that no names or thicknesses can be used. For this purpose it is sufficient to say the vein exceeds 5 ft. in thickness and carries a good roof and fair floor. It has three strata all differing in physical structure. The three layers are here designated as top, middle, and bottom benches.

	Top Bench	Middle Bench	Bottom Bench	Total Vein
Moisture, per cent	1.08	1.10	1.15	1.12
Volatile matter, per cent	32.40	32.40	31.26	32.10
Fixed carbon, per cent	61.40	63.60	59.40	62.54
Ash, per cent	6.20	4.00	9.34	5.36
Sulfur, per cent	.58	.73	1.47	.80
B.t.u. value	14,218	14,554	13,260	14,481
Ash fusing point, deg. F.	2,680	2,600	2,280	2,540

Before discussion let us fix four points in our minds. First, the producer wants as diversified a market and as high a price as the quality of his product warrants. Second, the consumer wants coal suitable for his requirements at the lowest figure. Third, with possibly one exception, the demand for the smaller sizes is increasing for all uses. Fourth, certain users desire a fuel fairly consistent in analysis.

A bed sample analysis would fix the above coal as suitable for by-product use and some business may be thus placed. The other analyses show that such business would probably be lost due to fluctuation of ash content in carload shipments. Those familiar with special use requirements would pick many flaws from the showing of this group of analyses.

If shipped as run-of-mine the trade would slowly, but surely, class this apparently good coal as average steam fuel.

Most operators screen in order to get the advantage of higher prices for the large sizes but the owner of the above coal, by intelligent operation, can enlarge the scope of his market and increase his earnings by so doing. If the cutting is done in the bottom bench the screenings will run too high in ash, and possibly sulfur, for the more exacting uses. By cutting in the top or middle bench the slack should be acceptable for any use in its class and size. The uses of the prepared sizes would depend largely on the relative firmness of the structure of the several benches. It may be possible to obtain a good water-gas size and a fair domestic lump.

It is possible, and entirely practical, to ship many coals, usually considered ordinary, as excellent products for the uses to which their common class may be put.

The writer has observed a number of instances where veins are constituted along lines somewhat similar to that pictured above. It may be one of the committee's questions to decide just how prevalent this condition is and whether or not veins are consistent in such respects over large areas.

Consumers have told many operators "the coal looks and analyzes all right but it just doesn't suit." The reason may be found in its component parts. One small stratum may contain most of the ash, sulfur, or phosphorus or so differ from the remaining coal as to make the whole undesirable for some purposes when loaded in disproportional volume.

The writer has examined many thousand analyses and some coals run remarkably consistent. However, it is surprising how many coals show great divergence in shipment. Veins running over 6 per cent. inherent ash have produced whole cargoes of sized coal running less than 4. Other commercial analyses show ash and sulfur contents far in excess of the average in the seam.

It would seem that the scope of the classification is much wider than would appear if it is to result in practical usefulness. Classification of coals, by beds and districts, from bed sections may ascribe virtues which coals do not have or condemn coals which may be made worthy.

INDEX

(NOTE: In this index the names of authors of papers and discussions and of men referred to are printed in SMALL CAPITALS, and the titles of papers in *italics*.)

*Bold face type indicates Vol. 76, *Trans.* A. I. M. E.

* Bold face type indicates Vol. 76, *Trans*. A. I. M. E.

S